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J. W. RICHARDS, PH.D., President.

JAMES H. MCGRAW, Vice-President.

CURTIS E. WHITTLESEY, Treasurer.

E. F. ROEBER, Secretary.

TELEPHONE CALLS, 4760 BRYANT.

E. F. ROEBER, PH.D., Editor.

H. C. PARMELEE, Western Editor.

J. M. MUIR, Manager.

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The Abrogation of the Hill Iron Ore Lease.

There are two reasons, each amply sufficient in itself, for the abrogation by the United States Steel Corporation of its lease upon the Great Northern or James J. Hill iron ore lands, the one being that the operation does not promise, from the present industrial position, to be profitable to the Steel Corporation, the other that the lease stands open to question under the Sherman law.

As to the extent of the ores, it may be said that there has come to be reasonable assurance that more than 300,000,000 tons will be found, of fair quality. None of the ore is of good quality, judged by the standard of other ores being mined. Practically all requires washing. The Colerine washer, installed by the Steel Corporation to treat the first of the Hill ores mined, cost to the end of 1910 the tidy sum of \$1,461,584.33, established to wash practically the first of the nearly 250,000,000 tons of ore which has thus far been taken out of the Mesabi range. The analyses of the Steel Corporation's Mesabi ores in 1910 were as follows as to iron content:

NATURAL, DRIED AT 212 DEG.

Group No. 1.....	52.9201	60.2171
Group No. 1 (high Mn).....	50.0105	56.6986
Group No. 2.....	54.4243	60.9739
Group No. 3.....	49.5229	57.0558
Group No. 4.....	47.2340	56.0731
Group No. 5 (D. M. & N.)...	50.4048	56.7034
Group No. 5 (G. N.).....	51.5741	57.7271
Group No. 7.....	46.4202	53.4499

Group No. 5, Great Northern Railway, may be taken as fairly typical of the best of the Hill ore. The royalty rate in the Hill lease is 85 cents for the year 1907, upon ore analyzing 59 per cent metallic iron when dried at 212 deg. The custom in the ore market is to sell ore upon a guarantee in the natural state, the base guarantee being 55 per cent for Bessemer and 51.50 per cent for non-Bessemer. The Hill lease rate advances and declines by 4.82 cents per unit for ore above and below 59 per cent, except that a flat rate of 30 cents is placed upon ore below 49 per cent, that being regarded as the limit for ordinary merchantable ore. For each year after 1907 the royalty rate advances by 4 per cent of the 1907 rate for the ore of iron content involved. The minimum to be mined (and paid for if not mined on the assumption that it would have been 59 per cent ore) is 750,000 tons in 1907 and 750,000 tons more for each succeeding year until 8,250,000 tons is reached in 1917, the minimum remaining at that rate thereafter. A quantity of ore, whether of Hill or other ore, equal to the year's minimum is to be transported over the Hill road, the Great Northern, at the rate of 80 cents, which has been the fixed rate from Mesabi range to Lake Superior ports for years.

Should the open rate decline, the reduced rate is to be paid the railroad, but the royalty rate is to be correspondingly increased, royalty payments being made to the three trustees who act for the holders of "certificates of beneficial interest" distributed in 1907, certificate per share, to the stockholders of the Great Northern Railway Company.

The lease provided that the Steel Corporation could terminate it upon Jan. 1, 1915, provided it gave notice not later than Jan. 1, 1913. Our understanding is that the Steel Corporation purposes to take advantage of this clause, but that, while nothing definite has been done thus far, there is opportunity for negotiation with the Hill trustees, and not impossible with the aid of the Department of Justice, looking toward an earlier termination.

The general impression seems to be that none of the Hill ores will be found to equal or exceed the base rate of 59 per cent metallic iron, dried at 212 deg., that only a relatively small proportion will fall within a couple of units of that iron content, and that the bulk will fall considerably below it. This alignment operates to depreciate very seriously the apparent value of the Hill ores, considered as an 85-cent royalty mining proposition, because the penalty rate of 4.82 cents per unit does not cover the increased cost of mining, transporting, and smelting the lower grade ores. It is, indeed, very much smaller than is represented in the trade custom in selling ore as that custom is to compute the unit price by taking the price at lower lake dock and adding a 60-cent arbitrary (assumed as the average inland freight rate), making the unit value at the present season's price on Mesabi non-Bessemer ore 7.96116 cents, and ores below 49 per cent iron content, natural state, are subject to penalties at greater than the straight unit rate. The season's price is \$3.50 for ore containing 51.50 per cent iron, natural state. Thus the lease is a less desirable one than were the ores actually to be mined rich enough to involve the payment of the base rate.

During its entire life the lease involves a constantly increasing royalty rate, at 4 per cent of the 1907 rate, equal to an annual increment of 3.4 cents on the 85-cent rate. For ten years it involves an annual increase in the minimum to be mined. In 1917 the minimum would be 8,250,000 tons, at \$1.19 for base ore, equal to a payment of \$9,817,500. Very onerous also is the provision that the ore or an equivalent tonnage shall be carried over the Great Northern road, at 80 cents a ton, for the Corporation has two roads of its own, and it is common report that the service is performed at a cost of not more than 40 cents a ton, so that here is a direct loss, 40 cents on 8,250,000 tons, of \$3,300,000 additional. In discussing this phase of the lease reference is frequently made to the possibility of the open rate of 80 cents being reduced. Such reduction would, however, neither increase nor decrease the Corporation's loss through the operation of this lease. In computing the advantages or disadvantages the payment stands in relation to the cost at which the Corporation could perform the service of carrying its ores on its own roads, and not in relation to the open rate charged to outsiders.

It is the common impression that the Steel Corporation entered upon the Hill lease for the direct purpose of preventing these ore deposits from falling under the control of existing or prospective competitors. In that sense, of course, the lease

may possibly be interpreted as having been in restraint of trade and perhaps under the ban of the Sherman law. It may be observed that many of the features of the United States Steel Corporation, viewed in the light of a "trust," are in line with those modern developments which are held to represent progress toward a beneficial form of socialism, but it is difficult to see how the Hill lease represented such progress. It would seem that the value of the lease to the lessors was created by the desire of the Steel Corporation to control the properties. There was a distribution, regarded purely as a gift, of 1,500,000 "certificates of beneficial interest" to shareholders of the Great Northern Railway Company, and the stock market value of these certificates at one time rose to above \$80, equal to \$120,000,000 for the entire issue, a valuation which had dropped to about \$75,000,000 prior to the announcement that the lease would be terminated. So it would appear that this value, if it may be regarded as a value, was created chiefly by an effort to monopolize, and shrewd lawyers' logic would be required to defend it as representing progress toward a better form of conducting industry.

Aside from this feature, however, it is pertinent to observe that no evidence has appeared of competition being restricted. The pig iron capacity of the United States in 1906, the year in which the lease agreement was made, was 25,000,000 tons per year. Partly through new erection by the United States Steel Corporation, but largely through new erection by outside interests, the capacity has been increased to between 33,000,000 and 34,000,000 tons at the present writing, with furnaces in course of construction sufficient to make at least 35,000,000 tons total capacity. Present demand and production, however, are at the rate of only about 25,000,000 tons. The excess is such as to preclude any important influence upon prices of pig iron or finished steel products through the existence of this lease, as compared with the condition which would have existed to-day had the Steel Corporation allowed the Hill ore lands to remain upon the market. The Lake Superior region is developed as matters stand to such an extent that it can easily ship 50,000,000 tons of ore, and quite possibly not far from 60,000,000 tons in a season whereas this season's actual shipments promise to fall below 35,000,000 tons, and no season has yet shown a movement of 44,000,000 tons.

The burden of the Hill lease would have been less onerous, through distribution over a larger tonnage, if the Steel Corporation's operations had grown as was expected in 1906. Its capacity has grown in the intervening five years, but its actual output this year is not 10 per cent in excess of that of 1906. Doubtless the expectation in 1906 was that its productive requirements would be doubled by 1917, whereupon the ore required to be mined in that year under the Hill lease would have been less than 20 per cent of the total.

On account of the annual increment in the minimum tonnage until 1917 and the indefinitely continued increment in the royalty rate, the Steel Corporation relieves itself of a large part of the total burden by a discontinuance upon Jan. 1, 1915. We may compute roughly that the total royalty payments in the period 1907 to 1914, inclusive, amount, at the nominal rates, to \$27,234,000, and the loss in freight payment, at 40 cents a ton, to an additional \$10,800,000, making about \$38,000,000. Royalties, though much smaller, would have to be

paid elsewhere if this ore were not mined, but, on the other hand, the development and mining of the Hill ore during this eight-year period are presumably more expensive than the cost would have been elsewhere with continued mining promised. The abrogation of the lease may be effected for an earlier date by arrangements which may yet be perfected. In any event the Steel Corporation will find itself in better condition from a business standpoint, while it is fairly clear in view of the attitude of the Government that it will find itself in a much more comfortable position from a legal standpoint.

The Power-Factor of Electric-Furnace Circuits.

Managers of central stations using an alternating-current system are unpleasantly familiar with the word "power-factor." It connotes to their minds the necessity of buying copper for their generators, distributing mains and transformers to carry the "wattless current" that does no useful service.

In electric furnace plants which are generally operated by alternating current the power-factor must be taken into consideration, and it is especially important to look at the power-factor problem from the standpoint of future increased size of furnace units. In small furnaces such as those which are used for experimental work, say, of 25 kw to 100 kw, the power-factor is high, being from 96 per cent in the first named capacity to 85 per cent to 90 per cent in the second named capacity. But as the wattage rating of furnaces increases, other things being equal, the power-factor decreases, hence the wattless current increases. Consequently we have an untoward state of affairs, quite unlike the case with thermal economy, which increases with the size of the furnace.

An electric furnace has been well called "a short-circuit under control." It has a very low ohmic resistance. The current flow is opposed by the magnetic "reactance" as well as the ohmic resistance. The power in a direct current is equal to the product of the volts and amperes. With an alternating current this product is spoken of as the "apparent watts" or when divided by 1000 as kilovolt-amperes and not as kilowatts. The reading of the wattmeter gives the "actual watts." The ratio of the "actual watts" to the "apparent watts" (which is unity under ideal conditions and otherwise always smaller than unity) is called the power-factor of the circuit.

In an alternating current, therefore, Ohm's law in its simplest form does not apply and a modified form must be employed. Here the amperes are equal to the volts divided by the "impedance." The impedance is equal to the geometrical sum of the resistance and the reactance. The "reactance" can be described as that physical quantity which opposes the current flow by virtue of the electromagnetic properties of the circuit. Now these very electromagnetic properties of the circuit or more properly speaking the "self-inductance" are proportional (1) to the ampere-turns, (2) the number of lines of force that are cut in unit time by circuit and (3) to the frequency.

Let us suppose that the third of these, the frequency, remains constant as it does in any commercial plant and that while the electric furnace increases in size the voltage remains constant; then the amperes and consequently the ampere-turns increase. Furthermore, the number of lines of force increase because they are generated by the electric current in the iron work that encases the firebrick of the furnace. It is plain, therefore, that

the "reactance" increases with the increase in size of the electric furnace. However, the resistance decreases with increase in size. Both the decrease of resistance and the increase of reactance tend to reduce the power-factor and therefore to increase the wattless current. Let us consider two electric furnaces of 50 kw and 100 kw capacities respectively. If the emf is 100 volts and there is a power-factor of 100 per cent, the current of the first is 500 amp and the second 1000 amp. The resistances are 0.2 ohm and 0.1 ohm respectively. If the smaller furnace had a power-factor of 96 per cent at 60 cycles it must needs have a current of 521 amp of which only 500 would be useful. But even if the larger furnace were carefully designed its power-factor might be as low as 80 per cent and so it would require a current of 1250 amp, of which only 1000 amp are useful. If the design were careless the power-factor might drop as low as 50 per cent. The current would need be 2000 amp. to produce 1000 amp. of useful current.

Let us interpret this in commercial language. Let the electrometallurgist who utilizes only 1000 amp. picture to himself the busbars, cables and secondary windings of the transformer to carry 2000 amp. and also the increase in copper necessary to carry the primary current and convert this into dollars and cents. The picture is not pleasing, especially when large capacities, say upward of 2000 kw, are needed.

There are several ways to mitigate this evil by proper design. One possible method is to reduce the frequency. But this means electrical machinery of special design and increased cost.

A more practical way (for usually the electric installation is already placed) is to increase the ohmic resistance by change of design of the furnace. Such a procedure results in the increase of the working voltage and of the power factor.

A third way is to increase the magnetic reluctance of the iron work of the furnace by change of design. For instance, it is possible to use large firebrick block 18 in. x 12 in. x 8 in. instead of the small 9-in. x 4½-in. x 2½-in. firebrick backed up by iron plates so often and so conveniently used. With this iron eliminated the magnetic circuit has a greater reluctance and so the reactance is diminished. It might be even possible to have the buck-staves, tie rods and facing plates made of some non-magnetic special steel. All these possibilities must be considered in designing a 1000-kw electric furnace.

We are glad to call the attention of our readers to this most interesting aspect of electric-furnace design. In this twentieth century it is necessary to attain the maximum of commercial efficiency. When viewed from the proper standpoint commercialism has a broad and ideal meaning and a sense quite different from that used by the poor muckrakers. There must be the application of the pure sciences such as chemistry, physics, and economies in the profession of metallurgy. These must be coupled with the use of arts. We have then a pleasing combination of the "knowledge why" and the "knowledge how." It is plain that this combination is needed in electric-furnace developments. We are especially glad to call attention in these columns to the subject of the power-factor of electric furnaces at a time when the electric furnace is invading the metallurgy of iron, steel, copper, lead and brass. For it is in these columns that all the first achievements of electric furnaces have been recorded, and electric-furnace pioneers have always been our readers, contributors, and friends.



The San Francisco Meeting of the American Institute of Mining Engineers.

(Special Correspondence.)

The Pacific coast has seldom offered such attractions as it spread before the mining engineers at this meeting. The meeting is, in reality, a fête, a festival of good things.

Starting from Chicago by special train on Sept. 30, the Eastern members, gathering in numbers as they sped westward, paid their first respects to Western scenery by halting two days at the Grand Canyon of the Colorado. The first day was perfectly clear, and the marvelous depths and details of the canyon stood out clear-cut and distinct; the second day clouds and mists, with here and there a rift and patch of sunshine, showed the great gorge in its most playful mood, and, to many, in its most artistic setting.

Los Angeles was reached on the afternoon of Oct. 5, and then followed a strenuous day and a half of unceasing pleasure and unstinted hospitality from the local members. Trolley ride to Pasadena, lunch on the ocean pier at Venice, fish dinner at Redondo and banquet at the Los Angeles Club were incidents of these days, while the local oil field at Bakersfield was also visited. Mr. Mullholland, of the Los Angeles Aqueduct staff, gave a fine illustrated lecture on this gigantic scheme, and its power possibilities. Mr. Comstock, head of the committee on local arrangements, proved a wonderfully efficient manager. A day at Santa Barbara and one on the cliffs at Del Monte or off to Santa Catalina island completed the sojourn in southern California.

Reinforced by the southern California members, the expedition reached the city by the Golden Gate on the morning of the 10th. Here the serious and formal work of the meeting began. The afternoon of the 10th and the mornings of the 11th and 12th were given over to sessions for reading and discussion of papers, sessions more than ordinarily interesting for variety and value of papers presented and vitality of discussion. The full program of these sessions was as follows:

TUESDAY AFTERNOON, OCT. 10.

- Chase, C. A.—"Notes on the Liberty Bell Mine."
 Aldrich, Jr., T. H.—"Electrolytic Oxygen in Cyanide Solution."
 Young, Geo. T.—"Slime Filtration."
 Hall, E. J.—"A Modification of the 'Gay Lussac' Method for Silver Bullion Containing Tin."
 Durham, E. B.—"Electrolytic Refining at the U. S. Mint, San Francisco."
 Daniels, J.—"The Fritz and Cox Laboratories of Lehigh University."
 Bahnay, L. W.—"Rapid Estimation of Available Calcium in Lime Used in the Cyanide Process."

- Lass, W. P.—"Cyanide Plant at the Treadwell Mines, Alaska."
 Christy, S. B.—"Progress in Electro-Amalgamation."

WEDNESDAY MORNING, OCT. 11.

- Requa, Mark L.—"California Oil."
 Janin, Chas. (Introduced by H. Foster Bain)—"Present-Day Problems in Dredging."
 Yale, Chas. G.—"Gold Production of California."
 Reinholt, Oscar H.—"Fuel Problems of the Pacific."
 Reinholt, Oscar H.—"Government Coal Mines in the Philippines."
 Dilworth, J. B.—"The Black Mountain Coal District, Kentucky."
 Discussion by J. Langdon.
 Locke, Augustus—"Notes on the Geological Structure of Tonopah, Nev."
 Catlett, Charles—"Phosphorus in Coking Coal."
 Porter, J. J.—"National Valuation and Quality Efficiency of Furnace Stock."

THURSDAY MORNING, OCT. 12.

- Nisho, K.—"Historical Sketch of the Mining Industry in Japan."
 Brunton, D. W.—"The Laramie Tunnel."
 Locke, Augustus—"Classification of Ore Deposits for Miners."
 Stevens, Blamey—"The Laws of Igneous Emanation Pressure."
 Stevens, Blamey—"Physical Data of Igneous Emanation."
 Irving, J. D.—"Some Features of Replacement-Ore-Bodies and the Criteria by Means of Which They May Be Discovered."
 Gamba, F. P.—"Geology of Some Mines in the South of Colombia, S. A."
 Keyes, C. R.—"Vadose Ore-Deposition in Arid Regions."
 McCollum, Burton—"Electrical Practice in Mines."
 Vallat, B. W.—"Geology and Operation of the Newport Mine."
 Read, T. T.—"Mineral Resources of China."

THURSDAY AFTERNOON (in the open-air Greek Theater at Berkeley.)

- Raymond, R. W.—"Reminiscences of the Beginning of the Institute."
 Becker, G. T.—"Biographical Notes on S. F. Emmons." Abstracts of some of these papers, and the discussions will follow in our next number.

The excursions arranged for the meeting were pleasurable and highly interesting. On the afternoon of the 11th, a special train left for Palo Alto, where over two hours were spent at the Stanford University. The next afternoon a similar trip was made to the University of California, at Berkeley, with a short session held in the famous Greek Theater.

Oct. 13 was given over entirely to seeing the gold dredges of the National Consolidated Company, along the American River



near Sacramento. The visit was a revelation of the latest triumphs of this manner of mining gold. The enormous new electrically operated dredge No. 9 was eating into *terra firma* at the rate of 12,000 cu. yd. per 24 hours. Digging to 55 ft. below water level, and taking down 10 ft. above, this enormous 2400-ton craft, operated by 1000 hp, was said to be working over earth carrying 8 cents worth of gold per cubic yard, at a total cost of not over 4 cents per yard, including depreciation and interest on capital invested.

Equally interesting was the effort being made to reclaim this worked-over ground. Working in a pond, the dredge leaves the gravel piled high as heaps of stones, while the soil or silt impregnates the lower layers under water. The most successful scheme seems to be the crushing of the stone for railroad ballast, road material and concrete. This crushed stone is worth 75 to 90 cents per ton at the crushers, and is being shipped all over central California. This disposes of the stones down to the silt or soil. The ground is then leveled off and planted with eucalyptus trees or rubber trees. These grow in the stony ground if silt or soil is not over 2 ft. beneath, and the tract of land becomes green again. An alternative would be to strip 2 ft. or 3 ft. of soil from the land before working, pile it aside and return it to the leveled-off ground. In any event, appropriate legislation should compel all the gold diggers to leave the country in a cultivable condition, as part and parcel of the expense of getting out the gold, and prohibit leaving the worked-over ground a stony waste, more barren than the desert of Sahara.

Returning from Natoma through the beautiful Sacramento valley, another sacrilege was visible from the train in the shape of a tall smelter stack vomiting out a poisonous cloud of white smoke which enshrouded the valley and mountains for miles away. Such a white pest, prejudicial to health, agriculture and natural beauty, should be suppressed with a firm, determined hand. Some practical methods of stopping the smoke are now known; it is simply a question of private greed *versus* public good, and may the public quickly win out!

Saturday, Oct. 14, 1911, is now memorable in California as the day of breaking ground for the great 1915 exposition. The President of the nation was on hand to start "making the dirt fly." Members of the Institute and guests were presented with grand-stand tickets, and joined the throng, 100,000 strong, which gathered in the stadium of Golden Gate Park. In bright sunshine, under a perfect sky, the surrounding hills green with verdure as a background, the tranquil Pacific visible to the west, and to the accompaniment of a military parade, martial music and echoing salutes of cannon, the first sod of the great exposition was turned. Just before, President Taft, in a ringing address audible to thousands, reviewed in brief the history of the canal, told them that 75 per cent of the work on it was now done, and predicted its completion by July, 1913. One sentence

of the President's address lodges in our memory as unusually forcible and strikingly true: "Had the sea-level plan of the canal been adopted it is altogether probable that no one who now hears the sound of my voice would have lived to see the canal completed."

Finally, Oct. 15 is now memorable in the remembrances of the visiting mining engineers for the day—one glorious day of throbbing life—in the sacred groves of the red-woods. Bohemia the spot was called. Here the Bohemia Club stopped the wood-cutter's ruthless axe and spared a spot in which to bury care, to worship in nature's temple. It is a small valley, the ground springy with its forest carpet, with scores of congenial spirits in the shape of our giant brothers, the big trees. Some are 10 ft. in diameter and over 300 ft. high. Among such we pigmies stalked and strutted and enjoyed our little day. A splendid orchestra added to the charm some lovely music of Bohemia, two of the numbers being conducted by the composers themselves. A more harmonious assemblage of artistic enjoyments, beauteous surroundings and congenial society cannot be imagined. Here the above photograph was taken.

On the evening of the 16th the California section of the Mining and Metallurgical Section met at dinner, with many members of the Institute as guests. Later the withdrawal of the public coal lands in Alaska was discussed pro and con, with considerable spirit. The wide diversity of opinion expressed showed how impossible it must be, in the nature of the case, to secure a uniform verdict on any such matter which would correctly represent the views of the society as a whole. While the discussions are certainly profitable and instructive and lend a proper interest to the meetings, yet a final expression of the opinion of the society as a whole could not be arrived at without great difficulty and arousing of bitter antagonisms. Such a practice is a principle of separation, a source of discord which would ultimately disrupt any society trying to put it into practice.

Tuesday, Oct. 17, saw the fortunate "100" leaving for their seven weeks' trip to Japan, and the more occupied and less fortunate remainder leaving for their homes. The trip to Japan, stopping at Honolulu, promises to be of absorbing interest. Dr. Kanda, of Japan, has come across the Pacific as the advance guard of welcome, and a glorious eighteen days of strenuous experience awaits the expectant engineers in the Kingdom of the Rising Sun—of which more anon.

J. W. R.

The Western Metallurgical Field.

Company Reports.

The metallurgical results obtained by El Oro Mining & Railway Company, Ltd., El Oro, Mexico, are made public in the last annual report for the period ending June 30, 1911. The total ore mined during the year was 360,504 tons as compared

with 317,824 tons in the preceding year. Changes have been made so that sulphide ores can be crushed and cyanided separately, thus improving results. Additions were made to plant consisting of six agitation tanks, two thickening tanks and three revolving filters. These were installed principally to aid in the treatment of sulphides and high silver ores.

The percentage recovery for the year was slightly lower than in the preceding period, but this was due to the fact that a lower grade ore was treated in larger quantities, thereby reducing the percentage recovery but making an increased net profit on operations. If the former high recovery had been maintained it would have resulted in a loss on the extra bullion produced. The principal data are in the following table.

Mill in operation, days.....	349.4
Tons crushed per stamp per 24 hours.....	10.3
Tons crushed	360.294
Gold value per ton	\$5.99
Silver value per ton	\$1.64
Per cent gold extracted	91
Per cent silver extracted	73
Per cent total extraction	87

The costs per ton for the year under consideration are as follows:

Mining	\$1.55
Development	0.74
Milling	0.17
Cyaniding	0.91
Water supply	0.02
General expense	0.22
Taxes	0.29

Total \$3.90

The Consolidated Mercur Gold Mines Company, Mercur, Utah, is apparently nearing the end of its life as a prosperous concern. The mines of the company have had a most excellent record for the past twenty-five years, but for the last three years it has been apparent that the grade of the ore was becoming lower and the profit less. The net profit in the last fiscal period was less than \$500, and this did not come directly from the mill but from outside sources. The average gross value of the ore treated was \$3.21, this including the value of old tailings retreated which amounted to 12 per cent of the total. The total quantity of material treated was 239,190 tons, and the value extracted was \$2.32 per ton. Loss in tailing was \$0.88 per ton. The cost of producing gold was slightly higher than its value, or \$20.9218 per ounce. This shows a loss per ounce of approximately \$0.25.

Of a more encouraging nature is the August report of the Goldfield Consolidated Mines Company, Nevada. It shows the largest tonnage ever produced by the mines in any one month, the total being 30,580 tons, or nearly 1000 tons per day. The average gross value was \$29.77 per ton, and the total net profit was \$634,658. Of the total mine production, 30,508 tons were treated in the mill with an average extraction of 94.75 per cent, or \$20.75 per ton. The total operating costs amounted to \$7.46 per ton. The company is now engaged in constructing a plant to treat both the accumulated and current residues from the present concentrate treatment process. Two Edwards duplex roasting furnaces, one Baker cooler, one 5-ft. by 18-ft. tube mill and other accessory apparatus are being installed in a steel building erected for the purpose. The additional tanks for collecting the roasted product and agitating in cyanide solution will be contained in an addition to the present plant. Tests show that an average of 90 per cent of the value of these residues can be recovered with an annual saving of at least \$80,000. The cost of the additional plant will be about \$70,000.

Renewed Interest in Mining.

The decisive steps taken by Western commercial and scientific bodies to renew public interest in mining are apparently bearing

fruit. From present indications it is certain that mining in general but ore treatment in particular is going to profit by the efforts being made to bring the operator and technical man closer together, and there can be no doubt that the general education of the public will result in attracting money for mining investment on a sane basis.

The latest developments on this line which give promise of improving general conditions are two, viz., the proposed unwatering of the downtown district of Leadville, and the meetings planned jointly by the Colorado School of Mines and the Colorado Scientific Society. Co-operation for mutual benefit is the underlying motive in each case.

The downtown district of Leadville has been unproductive for some years owing to the large volume of water encountered in the mines and the expense of pumping it to the surface. Drainage tunnels have not been feasible, and until recently all proposals for pumping have seemed impossible. The one link necessary to complete the chain of forces which make the plan attractive is the Central Colorado Power Company, which has ample power at its disposal. At a recent meeting held in Leadville the power company outlined a plan for the formation of a pumping company with sufficient capital to purchase and install the necessary machinery. The power company proposes to furnish most of the working capital, and it is expected that the pumping company will ultimately operate at a profit to be derived from a tariff placed on net smelter returns of ore mined as a result of the pumping. Naturally the power company has a selfish motive in proposing this plan, but on the other hand, the plan is such that no one will profit unless valuable ore is produced. It is not expected that any one will benefit at the expense of any other party concerned.

The most important matter is to secure the co-operation of the owners of the mines which it is proposed to unwater, and to receive assurance from them that if they do not wish to work their mines they will make reasonable concessions to leasers. Taken as a whole the plan is very encouraging, for it is known that ore exists in the flooded district, which was only abandoned when the expense of combating the water became unbearable. The power company has competent engineers at work on the details of the plan and it is expected that a definite scheme will be ready for presentation to the mine owners in the near future.

The other plan referred to, that of meetings under the joint auspices of the School of Mines and the Colorado Scientific Society, is expected to bring improved results not only in renewed interest, but also in improved technical work. The plan contemplates holding meetings in the different mining and metallurgical centers in the State at which a selected speaker will open a discussion on some pertinent subject. The discussion will bring out the views of local operators as well as visitors, and the exchange of opinions must undoubtedly do much good. The list of speakers already obtained comprises the best technical talent in the State. Experts on mining, mine management, ore treatment and plant construction will discuss their subjects in districts where they are most appropriate. This will bring the engineer in closer touch with the problems of the various districts, as well as familiarize the local operator with the ideas and experiences of the engineer.

Improvements at El Paso Smelter.

For some time past the American Smelting & Refining Company has been making additions to its El Paso (Tex.) plant, for the purpose of handling the copper concentrates from the Ray and Chino concentrators. Three new oil-burning reverberatories have been built and are probably now in commission. Hitherto the Ray concentrates have been going to Mexican smelters, but will now be diverted to El Paso. The new Ray smelter recently taken over by the American Smelting & Refining Company will not be ready for operation for some time to come, and until then the Ray concentrates will be shipped to El Paso.

Ray Central.

Engineers Weed and Probert have recently estimated the cost of building a 1500-ton concentrating mill at the property of the Ray Central Copper Mining Company, Arizona, and for permanently equipping the mine for large production, at \$1,500,000. They estimate a 75 per cent extraction in the mill, concentrating six tons into one, and a smelting charge of \$6 per ton of concentrate. Assuming the smelter to pay for 95 per cent of contained copper, less 2.5 cents New York quotations, and a 13-cent copper market, the high grade ore should yield \$5 per ton net. The cost of production should be about 7 cents per pound, exclusive of amortization and interest.

Flotation Process in Idaho.

It is reported that the Macquisten tubes installed in the Morning mill at Wallace, Idaho, are giving satisfaction and producing a good grade of zinc concentrate. Reference to this installation was made more fully in our issue for May, 1911, page 236. The material being treated by the tubes is tailing from Wilfley tables, and contains about 10 per cent zinc. Siderite is present to the extent of about 35 per cent of the ore pulp, and is the principal substance to be separated. Its specific gravity is 3.8, while that of blende is about 4. Thus there is little margin for separation by ordinary concentration methods, and the flotation scheme was resorted to as offering the best medium of separation. It is reported that the installation is turning out zinc concentrate containing 50 per cent zinc.

The Iron and Steel Market.

Bookings of orders for finished steel products in October showed no material change in the total as compared with September, and on the whole fell slightly below the production, through orders for rails and fabricated steel being light, while in practically all other lines bookings were up to the production. Output showed a slight gain over September, being thus better than in any preceding month of the half year. The increased outputs were in the main obtained without the putting of any additional productive units in operation, the month being normally one of heavy outputs per unit, on account of favorable weather conditions.

There has been no break in the monotony of the price decline. Plates and shapes have suffered a decline of about \$2 a ton, and steel bars and black sheets a decline of about \$1 a ton, while tin plate was definitely reduced 20 cents a box or \$4 a ton. While nominally open market conditions prevail in the tin plate market, there is definite leadership by the American Sheet & Tin Plate Company, the subsidiary of the steel corporation, which last year made more than 60 per cent of the country's total tin plate output, and its prices are customarily adhered to by the independent producers. Its reduction was made because there had been such a general decline in other steel products and because the shading on small lots for early shipment which had already appeared suggested clearly the impossibility of the old price being maintained for the new season's business. About this time of year the large can makers customarily place their contracts for the first half or the whole of the new year.

In the products outside of tin plates which declined open market prices prevail in practice as well as in theory. Of the country's sheet production last year the American Sheet & Tin Plate Company made less than 40 per cent. In these strictly open-market lines the declines have occurred gradually, through a given low price first being made in the case of a few exceptionally attractive orders, then upon less attractive orders, and finally upon the common run of orders.

Weighting the important finished steel products outside of standard section rails approximately in the order of their tonnage importance, the declines per net ton in the average price of finished steel products during the past four months have been as follows:

July	\$0.70
August50
September95
October90
Four months.....	\$3.05

The average decline of \$3.05 a ton in the past four months came upon a market which had already declined \$1.35 since the beginning of the year and \$8.05 since the late months of 1907, when the maximum of the past nine years was reached.

That the cost line has been closely approached in the present market is declared by all producers, and there is no reason to doubt the statement. Attention should be called, however, to the fact that as a rule the mills are accepting specifications of a very much mixed character, and filling them, by prompt shipment, at practically the minimum market prices. That mill costs per ton would be materially lower were operations conducted upon better selected and arranged specifications there can be no question. Buyers are insisting upon prompt shipment against all specifications accepted, and the mills appear to be powerless, on account of the competitive situation, to exact premiums in proportion to the undesirableness of the specification. A great deal is said in the market about specially low prices being made on orders involving "attractive specifications" and of course there is some selection, but in the final analysis specifications being filled result in costs much higher than in periods when mill capacity is strained and orders are so long in filling that judicious selection and arrangement can be practised.

The industry is operating at about 75 per cent of its full capacity, making pig iron at the rate of about 25,000,000 tons a year, approximately the average of the three-year period of great activity immediately preceding the panic of October, 1907. The independents are operating at an average of slightly less than 75 per cent, and the steel corporation at a slightly better proportion, but nearly 15 per cent of its entire capacity is engaged upon export material, the independents conducting very little export trade.

There is no definite promise in general industrial conditions of an improvement in iron and steel consumption in the near future. Purchases, production and actual ultimate consumption are all proceeding at an even pace. Stocks in the hands of jobbers and manufacturing consumers are at a very low ebb, and there is the possibility of present prices proving so attractive to buyers as to induce them to increase their stocks toward normal proportions. No movement of this sort is discernible, but many buyers are openly stating that the moment they observe a hardening tendency in prices, or a slight falling behind in deliveries by the mills, they will promptly place orders for much larger tonnages. A very small event may thus at any time lead to important consequences by way of heavier buying, producing a spurt not altogether healthy through actual ultimate consumption not being correspondingly increased. On the other hand, the approach of winter may operate to curtail consumption in a minor degree and lead to slightly decreased operations. In a word, tonnage is very heavy, prices unsatisfactory and declining, and the market decidedly uninteresting.

Pig Iron.

Sales of pig iron have been light as to tonnage, but a large number of small orders have been placed, and deliveries on contracts are very good. Buyers had contracted very conservatively. There is scarcely any forward buying, no advancing tendency being observed in the market, while occasionally slight declines occur at one point or another. The actual consumption of merchant iron is believed to have been measurably in excess of production during the past two months, effecting a slight decrease in the rather large stocks in merchant furnace yards. These decreases are chiefly at inactive furnaces. An event of considerable interest in October was a break in the price of Bessemer pig iron. This market, the scene only two

years ago of transactions in 10,000 to 25,000-ton lots, held at \$15, valley, for more than a year, and then broke early in October on a sale of 1,000 tons at \$14.65, valley, iron being later offered at \$14.50, valley, without any important sales being effected. The market stands quotable as follows: Birmingham, \$10 for No. 2 foundry; delivered Philadelphia, \$15 for No. 2 X, \$14.50 for basic; f. o. b. Chicago district furnaces, \$14 to \$14.25 for No. 2 foundry; f. o. b. valley furnaces, 90 cents higher delivered Pittsburgh; Bessemer, \$14.50; basic, \$12.50; malleable, \$13; No. 2 foundry, \$13.25 to \$13.50; forge, \$12.75 to \$13.

Steel.

An awkward situation arose in the sheet bar market regarding settlements for October and fourth quarter tonnages. Practically all the consumption stood covered by term contracts, providing for price adjustment monthly or quarterly in a variety of ways, and frequently at the open market, or a slight fixed differential below the market. There was, however, no actual market in existence. There was the tradition of the old price of \$22, f. o. b. maker's mill, Pittsburgh or Youngstown. This had not been broken in Bessemer steel, for which the term contracts almost invariably called, but open-hearth bars had sold in small lots as low as \$19.50, and the entire finished steel market had declined since the \$22 price on sheet bars had first been named. In these circumstances a condition of open competition developed, even though all consumers had contracted for their tonnage, and settlement prices were finally reached at a considerable range, \$21, delivered Pittsburgh district or Youngstown, being done in some cases, while in other cases lower prices, down to \$20.50, were made. The market is not precisely quotable, and we name the following quotations, which are more or less nominal: Bessemer billets, \$20, maker's mill, Pittsburgh or Youngstown; Bessemer sheet bars, \$21, delivered Pittsburgh district or Youngstown; open-hearth billets, \$19, and open-hearth sheet bars, \$20, both f. o. b. maker's mill and subject to frequent concessions.

Finished Material.

Strictly open-market conditions prevail in plates, shapes, bars, and sheets, and on these we quote minimum prices done on fairly attractive orders, these prices being occasionally shaded, while on the least desirable orders slightly higher prices are obtained; in wire products the market is maintained in Pittsburgh and the East, but is frequently if not usually shaded in the South, Southwest and West; in tin plates the market is steadily held. The Pittsburgh basis, as a universally controlling factor, has disappeared in several lines.

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.15 cents, Pittsburgh.

Shapes, 1.20 cents for beams and channels, 3 in. to 15 in., inclusive, zees, and angles 3 x 3 and larger, Pittsburgh.

Steel bars, base, 1.10 cents, Pittsburgh; 1.15 cents, Buffalo; 1.20 cents, Chicago; iron bars, 1.25 cents, Pittsburgh; 1.22 cents, delivered Philadelphia; 1.17 cents, Chicago.

Wire nails, \$1.65, base; plain wire, 1.45 cents, base, Pittsburgh.

Black sheets, 28-gage, 1.85 cents; galvanized, 2.90 cents; blue annealed, 1.40 cents, f. o. b. Pittsburgh or Mahoning Valley; 2 cents, 3 cents, and 1.50 cents respectively, Chicago; painted corrugated roofing, \$1.35 per square, and galvanized \$2.50 per square, Pittsburgh, but frequently shaded.

Tin plates, \$3.40, Pittsburgh, for 100-lb. cokes.

CORRESPONDENCE.

An Instructive Experiment with the Flow of Heat.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—The following simple experiment may be of interest, as it shows some of the peculiarities of the flow of heat through bodies.

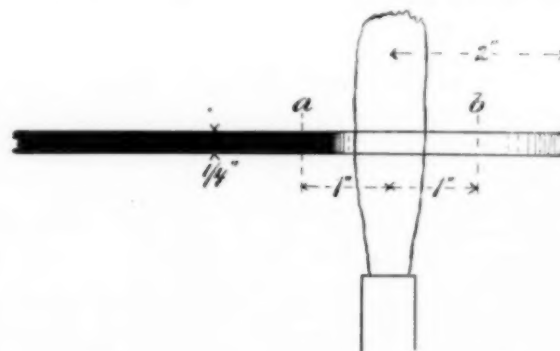
A copper rod $\frac{1}{4}$ in. in diameter was held in a Bunsen blast lamp flame as shown in the figure, the essential feature being that one end projected only slightly beyond the flame, while the other was practically infinitely long. When the stable state was presumably reached and the rod was at a bright-red heat in the flame, the visibly hot part was unsymmetrical, extending much farther toward the short end, in fact quite to the end of the rod.

This is shown in the figure, the white representing the red-hot part.

The heat enters the rod in the middle of the flame and flows from there to the two ends. It seems to follow, therefore, that more heat flows through the section *a* of the long end than through section *b* of the short end, both being equally distant from the middle of the flame; at *a* it was black and at *b* it was red-hot.

There is, therefore, less loss of heat to the air from the red-hot surface than from the cold surface.

It seems that the heat flowing to the hot end meets with a high virtual resistance at the surface of contact of the metal and the air, and that this resistance is in effect far higher than that of the metal. It would therefore seem to follow that if one side of a tank containing a red-hot liquid were made of a thin copper sheet, while another side was formed by the end of a long massive bar of copper, hence very thick, less heat would be lost through the thin wall than through the thick one.



ARRANGEMENT OF EXPERIMENT ON FLOW OF HEAT.

Or, in other words, the loss through the walls per square inch would be greater than that from the exposed top surface of metal.

If this were also true of the materials which are generally used for furnace walls, it would seriously upset our present ideas that a thick wall insulates better than a thin one. But a repetition of this experiment, using a rod of clay (the stem of an ordinary clay pipe) of the same diameter, in place of the copper, fortunately showed no such dissymmetry. A more carefully made test would probably even have shown a dissymmetry in the other direction.

A carbon rod acted like the copper one, but the dissymmetry was far less.

This apparently very high surface resistance to the flow of heat may be similar to the one existing in steam boilers, the presence of which can be shown by the fact that a postage stamp pasted on the fire side of a water tube will not even be charred by the flames. This high-surface resistance could be made use of in furnace walls in the form of joints between bricks.

The virtual resistance at surfaces seems to be high as compared with that of copper, but may be low as compared with that of the usual heat-insulating materials. Existing data on this subject seem to be deplorably meager, and perhaps quite unreliable in some respects.

If the thermal resistivity of copper decreases greatly with an increase of temperature then the above results might be explained quite differently.

CARL HERING.

Philadelphia, Pa.

Reminiscences of Robert Wilhelm Bunsen.

By DR. HANS GOLDSCHMIDT.

The centenary of Bunsen's birth has given me as an old pupil of the great savant, the idea of writing down some reminiscences of my student days at Heidelberg.

Of the many who have taught me in school, at home and at the University, Bunsen is the one teacher who impressed me most deeply as being truly a teacher by nature.

All who worked in Bunsen's laboratory had every possible opportunity of learning every detail of the science of chemistry from the great master, not only theoretically, but also from the purely practical, manual point of view.

Of course, Bunsen only devoted himself in this way to the regular and diligent worker in his laboratory. He was always ready, however, to help the occasional student, and was never otherwise than friendly to the less diligent; but the student who did not work at the laboratory regularly, could not but feel that Bunsen preferred to give his time to those who worked hard.

He put himself at the disposal of the latter out of working hours as well; indeed, I always had the feeling that Bunsen liked to be "bothered" by those students who were in earnest in their work. Strange as it seems, the constantly repeated monotonous teaching never seemed to be a torture to him.

I am speaking too of a time when Bunsen was fully 75 years old. I was, therefore, among the last of his pupils.

Bunsen attached special importance to his pupils being thoroughly conversant with qualitative analysis; he only allowed experiments in quantitative analysis when he knew that the student was complete master of the former.

As in other laboratories, so in Bunsen's, certain numbered mixtures were given out to be analyzed. He had the results put down in a small note book which he always carried about him. I shall never forget his delightful remark, when I asked him to tell me if the results of the qualitative analysis which I put before him were correct. I named the three or four elements which I had found, and the old gentleman gave me a glance full of meaning and said: "I don't want to know what you have found, please tell me what you have *not* found." He then went through the different groups of elements with me and made me describe exactly the process of analysis. I had to explain in each case how the reactions had taken place, and had furthermore to write down the respective chemical equations and put down separately what I had seen, and what I had not seen. From this it can be seen how seriously the great savant dealt even with beginners.

Bunsen was never tired of showing his students even the smallest secrets of the science; how a cracked test-tube can be made fit for use again by the blast lamp, how glass filaments are drawn, how a filter is correctly placed in the funnel, and many another useful piece of knowledge.

The tests known under the name of "Bunsen flame-reactions" were of very special interest. Even at his advanced age and in spite of a somewhat trembling hand, Bunsen could carry out these fine microscopical chemical operations with a skill and a perfection that excited wonder.

The palm of the left hand of the experimenter played an important part in this. Bunsen used it in this way: he rubbed the ingredients into it with the help of a penknife; during the

experiment, the fold of the hand made by closing the little finger over the palm, formed the provision store for the mixture, used in various ways during the analysis. The thumb and the first and second fingers of the left hand remained free for use, in spite of the closed little finger.

These researches, which were carried out in the flame of the Bunsen burner, took the place of and were an improvement on the so-called blowpipe analysis, which Bunsen did not like. It was his capacity for obtaining the most striking reactions with the smallest means, that marked Bunsen's teaching and was of the greatest value didactically.

Those who wished to take their degree were not allowed by Bunsen to work out a doctor-thesis, the viva voce examination was enough for him. His point of view was that the young student was not in a position to do independent work, and that, therefore, the doctor-thesis could only be due to the initiative of the teacher; he, therefore, thought it better for the student, as long as he remained at the university, to deepen and widen his knowledge thoroughly in all the fundamental parts of the science placed before him, without trying to break out new ground himself.

As an examiner, Bunsen was particularly thoughtful and

kind. Every one who had listened to his lectures attentively and worked through them thoroughly, was certain to get a good place in the examination. In my examination, he questioned me specially on his iodine titration, which he had gone into with me in the laboratory with special care so that he could not but be certain that I had it all at my finger-ends.

I may be allowed here to repeat a remark made by Geh. Rat Kopp and which was current among the students in those days—*si non è vero è bene trovato*—"Whoever passes his examination in Heidelberg with rite (the lowest class), has it written down in pen and ink that he is an ass." Little Geh. Rat Kopp, who is best known through his great work on the "History of Chemistry," was the faithful companion in walks, and above all on journeys, of the physically much bigger Bunsen.

Bunsen was pleased if any of his pupils, even after the examination,

continued to work under him on subjects of inorganic chemistry, which he had worked at himself. I myself, for instance, seized the opportunity of working at the production of calcium and strontium from molten chlorides, by means of electric current.

In those days this was a hard piece of work, as it had to be carried out with a Bunsen battery, and the only electrical measuring instrument was a tangent galvanometer. Not only was it hard, but particularly hot work, and very troublesome, because the chloride had to be melted on an open charcoal fire.

In spite of the heat of summer, Bunsen insisted on watching the experiment for hours, even long after the closing of the laboratory. He even considered it necessary to be present at the separating of the crucibles, in order to see and control the results.

He then went carefully through the theoretical calculations with me and took care too that the small reguli of calcium and strontium (which I still possess) should be fused in petroleum free from water, which he gave me himself out of an old store. He showed me also the process of fusing in the small tubes.

On these occasions the old gentleman sometimes became very talkative and would converse on matters that lay far outside



the realms of chemistry. He liked to talk about his journeys. A visit to the island of Majorca had left particularly pleasant memories behind.¹

Another time he told me stories of his teaching days at Göttingen, even then fifty years back. Among other things, he told me of a friend who, in consequence of a bet, had deciphered an Assyrian inscription within three weeks. These old memories seem to have struck particularly deep root in him.

Of greatest interest to me was a remark Bunsen once made, when he was discoursing on the advanced technical knowledge of the Egyptians: he said he had been shown small ancient copper bowls from the old wonderland of the pyramids, which by their form had suggested to him the idea that they must have been precipitated with the help of the electric current. This bold suggestion from a man of very sober thought is very striking.

I remember a particularly interesting occurrence when Bunsen allowed me to make some spectral-analytical experiments, such as were often made in his laboratory by the students. The experiment consisted chiefly in this, that the student, who had at his disposal a considerable number of pure substances in small glasses, attached a little of one substance to a platinum wire by dipping it into that substance, and then introduced it into the Bunsen flame, observing the rays or lines in the spectroscop and putting them down on a graduated scale.

The sodium line served as basis. Bunsen would point this one out with a cigar he held in the flame. As is well known, he was a smoker of strong Havanas. He never failed to point out that in every Havana you could see clearly also the red lithium line. If, by chance, a student himself had a cigar which showed no lithium reaction, Bunsen can scarcely ever have missed the opportunity of making a little joke, by pointing out that the student's cigar was no "Havana."

Among the elements I had to analyze were small quantities of the salts of caesium and rubidium discovered by Bunsen.

When Bunsen saw the lines I had put down for caesium, he shook his head doubtfully and expressed his opinion that it was impossible for me to see the two lines in the violet. The test was repeated over and over again, but I always put the lines exactly in the same place.

I never saw the old gentleman so excited and interested as on that occasion. He thought I must have made a mistake, for both the lines observed by me belonged to rubidium. The caesium given me to work with had not been perfectly pure, it contained traces of rubidium.

I could see these rubidium lines clearly, a fact that led Bunsen to declare that my eyes were exceptionally sensitive to violet rays, and he added: "Had I had your eyes, the separation and above all the discovery of rubidium would have been rendered much easier to me."²

He then invited me to go with him to his private laboratory. He had there larger quantities of salts coming from the Dürkheim mother liquors which contained only infinitesimal portions of rubidium and caesium.

He wanted to try if I could see the lines in this extreme dilution, though he scarcely thought it possible. And, as a matter of fact, it was impossible to recognize the salts in the substance handed to me by Bunsen. He made certain once more, however, that there was no mistake on my part in what I had seen and put down before in my earlier tests.

The last work Bunsen gave me was purely preparative and related to the preparation of cadmium borotungstate. The solution of this salt with its very high specific gravity was to be used for the separation of minerals.

As a rule, Bunsen was to be met among his students in the

laboratory for some hours both in the morning and in the afternoon. During the summer months, he was often even before his first lecture at the disposal of those who wished to learn from him his gas-analytical methods.

Besides this, he gave a daily lecture on experimental inorganic chemistry, a lecture which was probably unique. In one semester he gave a complete survey of inorganic chemistry and what he demonstrated and expounded was from beginning to end his own work, his own experience, his own research.

Bunsen attached the greatest importance to the illustration of his lectures by means of experiments. I cannot remember the failure of a single one. They too were carried out with the simplest means. For instance, Bunsen showed the working of the geysers with a simple apparatus, and prepared before his hearers electrolytically metallic lithium in a porcelain bowl.

One experiment was especially effective: oxy-hydrogen gas was introduced into a small china bowl full of soap and water. Soap bubbles filled with this gas formed at once on the surface. If a light were applied to them, they exploded and the bowl broke in pieces. Bunsen then repeated the experiment by forming the soap bubbles filled with the gas on the palm of his hand. The explosion was quite as loud, but the hand remained absolutely unhurt, because it formed an elastic foundation, not a hard one like the bowl placed on the lecture table.

After such experiments, the students' applause was particularly stormy, it expressed itself in noisy stamping. Clapping of hands was proscribed. These student ovations were also given to Bunsen when he entered and when he left the lecture room, and he always thanked with a friendly glance and a scarcely perceptible nod.

Bunsen's manner of lecturing may be described as classic, in the best sense of the word. His sentences were short, his delivery clear, his voice even in advanced age sonorous and fairly strong. He despised every kind of hollow figure of speech or brilliant comparison. In this respect one could find no greater contrast to him than the orator and philosopher Kuno Fischer, who was lecturing at the Ruperta-Carola, the Heidelberg university, at the same time as Bunsen.

Bunsen's favorite study was neither chemistry nor physics, it was geology. In his lectures he often referred to this branch of science.

But Bunsen was also in possession of more than ordinary knowledge in zoology, botany and above all, in medicine,³ so that it is not too much to maintain that he belongs to the men of his age who could claim the greatest general scientific culture.

Bunsen was of a meditative nature with an indubitably highly developed inner life. And this explains his dislike to ostentatious display, his wish not to appear in public or be disturbed by social duties more than was unavoidable. He was perfectly natural in his whole manner of being and acting. He was so thoroughly absorbed by his work, so taken up by it, that he felt no desire for things that seemed to him paltry.

With him, one always had the feeling that he placed the problem itself above everything and all that was personal as far out of sight as possible. That was why in his lectures, one never heard the expression: "I have found" or "I have discovered." He always spoke with the impersonal "one." Whether he spoke of the Bunsen burner, of the Bunsen battery, of the Bunsen valve, or of the spectral analysis-inventions that bore his name already while he was yet living—he never put himself forward, never mentioned his own name.

Bunsen was a great, even enthusiastic lover of the beauties of nature; he preferred wandering about on foot, so that he might gain strength in the fresh open air. Heidelberg, with its wooded surroundings, had become very dear to him. There is no doubt that he was deeply attached to his German Father-

¹He had gone there with the zoologist, Dr. H. A. Pagenstecher, professor at the University of Heidelberg. The latter has given a clear and charming account of this journey in a small book, which is still worth reading, "The Island of Majorca: Sketches of a Journey," by Dr. H. A. Pagenstecher (Leipzig: W. Engelmann, 1867).

²Cp. on a fifth element belonging to the alkali group. By Bunsen. *Berichte der Akad. der Wissenschaften z. Berlin*, 1861, I. 27.3/5.

³I may point out here that Bunsen owed his invention of the well-known contrivance the "Bunsen valve" to his medical knowledge of the flaps or valves in the veins.

land, through which he had wandered much in earlier years, but he was free from any onesided overweening appreciation of German intellectual efforts. His views and opinions were not limited by a narrow circle. Our planet did not suffice for his chemical and physical research, the eye of this gifted searcher pierced to the furthest worlds.

Much has been said about Bunsen's great modesty. Since Goethe declared that "only rascals are modest" this quality requires a certain reservation, when it is attributed to a great man. And this is specially called for in the case of our savant. He was an exceptionally high-minded man, most courteously kind in intercourse with others. That was noticeable even in small things. For instance, when a student entered his private laboratory with him, he would never go in first, but obliged the student to do so. If any one met the old gentleman out for a walk, he had to be careful to be in time in bowing first, for Bunsen never waited till he was saluted, but would take off his hat a long way off to the student whose face he recognized.

But together with this well-bred politeness, this simplicity in ordinary intercourse, there was in him a great, one may confidentially say, a too great consciousness of his merits. For Bunsen believed in, and set a value almost exclusively in, what he himself had seen, investigated, and discovered. Others' work he considered doubtful as a matter of course, very often ignored it and even treated with scorn many an investigator who did not deserve such treatment. He expressed this scorn, of course, only in incidental remarks and conversations, for he never wrote a controversial paper. In this depreciation of others' work and the careful avoidance of printed replies, he resembled the great Linnaeus, who, however, unlike Bunsen, took special delight in being himself in the foreground.

As an example, I may quote Bunsen's drastic remark—repeated to me by his late assistant Pagel, who was many years with him—on the periodical system of Mendeleeff, who was in Heidelberg from 1859 to 1860: "If a clever mathematician takes the Stock Exchange column of a newspaper, he also can discover fixed series of values that have no connection with one another." Bunsen never mentioned Mendeleeff's system in his lectures.

The great investigator also stood aloof from the then modern views on atoms and their groupings. His friend Kopp tried to enlighten him in an exceedingly clever and witty publication "From the world of molecules," which first appeared anonymously. This little book can be warmly recommended to every chemist who has a sense of humor, as far as that can enter into purely scientific questions. Often a very pertinent doubt on the part of the author pierces through the explanations, and this must have made the small work still more savory to the doubter Bunsen.

As Bunsen lectured substantially only on what he had himself investigated, so in his laboratory he only allowed work on subjects he suggested himself, and on which he had worked himself. As soon as a laboratory student chose a subject on his own account, the master's interest ceased. This peculiarity finds its explanation in his conscientiousness as a teacher; he wished to demonstrate and teach in his laboratory only what he was thoroughly acquainted with himself.

There were already a great number of Bunsen anecdotes current during his lifetime. Many of them were meant to prove the helplessness of the savant in things which were likely to be unfamiliar to him, especially as a bachelor, or which showed his awkwardness in society in general. The absent-mindedness attributed to every German savant played its part too in these anecdotes. If we were to trust these traditions, even only in their general tendency, our mental picture of the great man would be a gross caricature. Bunsen was a thorough man of the world, who had traveled a great deal and had associated with men of every condition, high and low. He knew perfectly well how to behave correctly and befittingly

in every situation in life. On the other hand, the great savant possessed a delicate sense of humor and a great taste for irony, both of which he often turned against himself; he liked to be ironical about himself. Probably those often foolish Bunsen stories owe their origin to this peculiarity of his, his joke having been misunderstood.

The domain of chemistry, especially of inorganic chemistry, that Bunsen was master of, was extraordinarily wide, for he possessed it in all its smallest details, whether it were a problem of the separation of rare earths, of the group of platinum metals, of the most difficult gas analyses, or of smelting processes of every kind. The work he did, as pioneer, in almost every department that he touched upon, was so comprehensive and always so thorough, that he did not seem able to find the time to make himself sufficiently acquainted with the realms of investigation of his contemporaneous colleagues. Thus he had not many collaborators; the best known are Kirchhoff and Roscoe.

To us, who look back, he stands out in solitary grandeur, far above others. His weakness, in ignoring the work and success of others, became finally a source of strength. His immovable trust in his own creative power absorbed him so exclusively, that he gave the world a great deal of new and most valuable knowledge, a fruitful and priceless possession for all time.

Essen-Ruhr, Germany.

Proposed Treatment of Sulphide Ores with Nitric Acid, or So-Called Rankin Process.

Editorial Correspondence.

The difficulties attending the economic treatment of ores containing the sulphide minerals of several metals have resulted in classifying such ores as refractory. This term is generally taken to mean that these ores do not yield readily to known methods of treatment, and consequently the newer developments in metallurgy are heralded as means of successfully treating refractory ores.

The proposal to treat mixed sulphide ores with nitric acid is an endeavor to solve this problem in an economical way. The recovery of the nitric acid is a vital feature of the process. In its present state of development it represents the original idea of Mr. H. D. Rankin as modified and improved by his co-workers.

In the following description of the process reference will be had to the treatment of an ore containing some form of the sulphides of iron, copper, lead and zinc, with accompanying gold and silver, either native or combined. The statements given represent the claims for the process as outlined to the writer by Mr. E. H. Westling, metallurgist for the company controlling the process. Experimental work has been done at Pittsburgh, Pa., and Lake City, Colo., and it is announced by the company that a commercial plant is to be built and operated immediately at Chloride, Ariz.

With the exception of gold, the products of the process are not metals ready for refining, but salts of the metals equivalent to high-grade ores suitable for reduction by present processes of smelting. In some instances the precipitated salts may find commercial use as such, as in the case of iron and copper, but for the most part they must be reduced to metals. Briefly, the primary products of the process are: Gold in the form of amalgam, silver as silver chloride, iron as hydroxide, copper as basic carbonate, zinc as hydroxide and lead as sulphate. Thus the process is designed to effect a chemical separation and concentration of the valuable constituents of an ore in a more economical manner than might be done in other ways.

A number of objections and queries will at once arise in the mind of the investigator as to the feasibility of the process. The complete precipitation and washing of salts in large quan-

¹Aus der Molekularwelt, published by Winter in Heidelberg.

tities; the economical regeneration and recovery of nitric acid; the completeness of the various reactions; the operation of a process with a gas under pressure; the economical operation of a process combining many separate operations; and the recovery of unfinished products which must stand further cost of reduction and refining—all these are points which call for proof by commercial operation on a large scale. The sponsors of the process, however, feel sufficiently assured on these points and claim that the method can be operated profitably.

Description of Plant.

The process naturally divides itself into three parts: The decomposition of the ore, recovery of the metallic salts and recovery of the nitric acid.

The decomposition of the ore is effected in a digester of cast iron lined with an acid-resisting mixture. This piece of apparatus is egg-shaped and supported in a vertical position. Within is an air lift agitator made of aluminium, connected by aluminium pipe with an air compressor outside. The proportional dimensions of air lift and digester are about the same as in a Pachuca tank. Above the digester, and tightly connected with it, is the ore bin, which can be tightly sealed. In the connection between digester and bin is a revolving cup feeder. In a suitable position above the digester is a receptacle for nitric acid.

The recovery of the metallic salts is accomplished in a series of precipitating and settling tanks, with such filtering and washing apparatus as may be needed. Ordinary amalgamating devices are provided for gold recovery, and a concentrating table is used for the separation of lead sulphate from the gangue.

The acid recovery system consists of a series of condensers and water towers. Aluminium pipe is used in all connections and cooling coils, and the tanks and towers are protected with acid-resisting mixtures wherever exposed to the acid gases. The water towers are constructed of paraffined redwood or cypress and filled with clean quartz. The first of the three towers is half full of water and the others are two-thirds full. At suitable places in the aluminium pipe are two indicators to show the condition of the gases at those points. These indicators are glass sleeves surrounding openings in the pipe, and the first is placed ahead of the first water tower and the second beyond the last tower.

Treatment of Ore.

Although the treatment of ore and recovery of acid are simultaneous operations, they will be described separately for the sake of clearness.

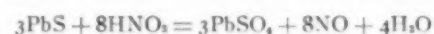
The process is worked intermittently on successive charges of ore and acid. Six tons of ore per charge is a convenient unit of capacity, and the time of treatment of each charge is between two and three hours.

The ore should be ground to about 30-mesh size, and contain at least 2.5 per cent to 3 per cent sulphur. If the sulphur content is below that figure the ore should be concentrated before treatment. At least 5 per cent sulphur has been found necessary if the reaction is to proceed without the application of heat. With 5 per cent or more sulphur the reaction with nitric acid generates sufficient heat for complete decomposition of the ore, but with less than 5 per cent it is necessary to heat the mixture. This is usually done by agitating with live steam instead of air for five or ten minutes at the beginning and end of the treatment.

The acid is run into the digester in quantity proportional to the amount of sulphur in the ore. It is claimed, however, that, if pyrite is present in considerable quantity, 5 per cent less than the theoretical amount of acid can be used, as the ferric salts act as catalyzers in oxidizing the sulphides to sulphates. It is customary to use acid of 1.15 to 1.42 sp. gr. The recovered acid has a sp. gr. of 1.25 to 1.35 and is used at that strength.

With the necessary charges of ore and acid in the sealed

ore bin and digester respectively, the feeder is started which slowly delivers the ore in small quantities into the acid. At the same time the air agitator is started and the feeding and agitation continue until the sulphide minerals have been converted into sulphates, all of which are soluble excepting that of lead. Some sulphuric acid may be used with the nitric if the ore contains lime, or to prevent the formation of basic iron sulphate. It is claimed that the conversion of sulphides to sulphates is practically complete, only about 2 per cent of the nitric acid remaining as nitrates of the metals. The following equation is typical of the reaction on which the process is based.



Throughout the treatment of a charge of ore the digester and acid recovery system are under about 25 to 30 lb. pressure. When the reaction is completed the digested mass is drawn into a tank where the solution is separated from the gangue by filtration or decantation. The residue is washed and passed over an amalgamated plate and Pierce amalgamator for the recovery of the gold. The lead sulphate is then separated from the gangue by concentration on a Wilfley table, the sulphate flowing off as a slime and collecting in settling launders. Naturally it will be contaminated with the fine gangue formed in crushing the ore, but it forms a high grade lead product for smelting.

The metals in solution are precipitated successively in a system of tanks, and the precipitated salts are washed and dried. Silver first is precipitated with salt, giving silver chloride which must be reduced to metallic silver. Iron next is precipitated with pulverized limestone (30-mesh) giving a fine granular precipitate of ferric hydroxide which settles rapidly. This precipitation is carried out at a temperature of 55 deg. C., which is about the normal temperature of the solution at this stage of the process.

A remarkable claim is made in connection with the precipitation of iron and the subsequent precipitation of copper. Ground limestone is used as the precipitant in each case, but the temperature is 20 deg. C. higher for copper than for iron. It is claimed that at 55 deg. C. limestone will precipitate iron without copper, and that upon raising the temperature of the solution to 75 deg. C. the same precipitant will throw down copper as basic copper carbonate. Wherefore, after the precipitation and separation of iron the solution is heated by passing live steam through it until a temperature of 75 deg. C. is obtained, when copper is precipitated.

The iron precipitate has no value unless circumstances should warrant its use as a paint pigment. A market might be found for the copper precipitate as such, but in all probability if any considerable quantity were produced it would have to be reduced to copper.

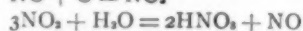
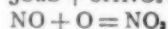
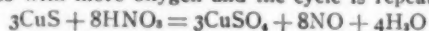
With iron and copper removed the zinc is precipitated with milk of lime, giving zinc hydroxide which is to be calcined and smelted. The zinc precipitate will be free from iron but may contain some manganese. It would seem that, inasmuch as lime in some form is the precipitant for iron, copper and zinc sulphates, certain amounts of calcium sulphate must contaminate the precipitates. Calcium sulphate is soluble at ordinary temperatures in about 500 parts of water, the solubility increasing slightly until a temperature of 100 deg. C. is reached, when it decreases rapidly. Hence the salt will precipitate unless enormous volumes of solution are used.

The precipitating tanks for these operations are fitted with pipes for air agitation, bronze pipe being used in all precipitations where copper is still in solution. The entire precipitation system is preferably arranged in duplicate to facilitate cleaning up.

Nitric Acid Recovery.

The principle upon which the recovery of nitric acid is based is shown in the following cycle of reactions. As shown before, the treatment of a sulphide with nitric acid gives nitric oxide

as one of the products. Nitric oxide and oxygen unite to form nitrogen peroxide, which is soluble in water at ordinary temperatures, forming nitric acid and nitric oxide. The latter then combines with more oxygen and the cycle is repeated.



The oxygen necessary for the reaction is supplied by the air used in agitating the charge of ore and acid. The nitric oxide and oxygen are constantly being forced out of the digester through the acid recovery system, the oxygen being supplied in sufficient quantity so that the gases escaping into the air at the end of the system will show an excess of at least 2 per cent O. It is claimed that 95 per cent of the acid used is recovered.

The gases first enter a large narrow cylinder having about two-thirds the volume of the digester. The function of this so-called mixer is to allow time for thorough mixing of the gases and formation of nitrogen peroxide. Some nitric acid is formed here by reason of the fact that when the reaction in the digester is well under way steam will issue with the other gases, giving the right conditions for the formation of some nitric acid. It is claimed that about 10 per cent of the acid is recovered here.

The gases are next cooled in an aluminium condenser or cooling coil, where another 10 per cent recovery is made. Following the cooler is the first water tower, through which the gases bubble and where the greatest part of the recovery is made, amounting to 60 per cent. Between the cooler and first tower is placed an indicator, where the gases should show reddish brown in color. Following the first tower is another cooler where about 3 per cent of the recovery is made. Finally, the second and third towers make about 12 per cent recovery. The acid drawn from the mixer and coolers is 1.25 to 1.35 sp. gr., while that in the towers gradually builds up to 1.35 to 1.40 sp. gr. before it is drawn for use. The first tower builds up in strength most rapidly and when its contents are withdrawn they are replaced by those of the second tower, and so on. The gases issuing from the last water tower pass through a second indicator, where they should appear colorless if the system has been working properly.

The cost of chemical treatment is determined by the quantity of sulphur in the ore. The acid loss is claimed to be only 5 per cent, and the precipitants used are inexpensive. In addition there will be the usual cost for preparation of the ore, pumping and agitating solutions, filtering, drying and marketing the products.

The results of the first attempt to operate the process on a commercial scale will be awaited with interest.

Fiber Testing Plant for Paper Manufacture.—Construction is under way on the paper machine house to form a part of the new laboratory of Arthur D. Little, Inc., engineers and chemists, Boston, Mass. This miniature paper mill will be L shaped, consisting of two rooms measuring approximately 25x35 and 25x52. Being located in the park system of Boston its exterior will be made especially attractive with brick and stucco walls and tiled roof. Although the entire equipment will be a miniature with a capacity of only 5 to 10 per cent of that of an ordinary machine, each piece of apparatus will be perfect in itself. The combination cylinder and Fourdrinier machine which is being made by the Pusey & Jones Company, Wilmington, Delaware, will be constructed along the same lines as the machines made by this company for the Forestry Service and the Bureau of Standards, but will differ somewhat in the method of driving as well as in the number of calendar rolls and suction boxes. Its length over all will be about 44 feet. The balance of the equipment will include two 50-lb. beaters with washers, pulp washer, Jordan, soda, sulphite and rotary digesters, steam generating plant, bleach dissolving system, vacuum pump, stuff chests and pumps.

The Grönwall Steel Refining Furnace.

BY THOMAS DUNCAN ROBERTSON.

The development of the electric furnace for iron and steel manufacture is comparable in its rapidity to that of the automobile or the aeroplane. In 1904 there were only four electric furnaces in Europe, while to-day there are over one hundred working in different parts of the world, some reducing iron ores, the majority, however, being used to melt or refine the products of older metallurgical processes and produce the material generally known as electric steel.

It is only natural that many types of electric furnaces have been proposed from time to time; some have existed on paper, some in the experimental stage and others have made iron and steel on a commercial scale. The latter class is generally subdivided into induction furnaces and arc furnaces. The former, it would appear, are on the decline, although they are fairly efficient melting machines when of small size, yet it is found to be very difficult to accomplish with them any appreciable refining. On the other hand, in the arc furnace cold materials are easily melted and the fluid charge can be refined to a degree of purity unequalled by any known process used in the iron and steel industry.

At the present time several types of arc furnaces have proved themselves capable of making good steel economically.

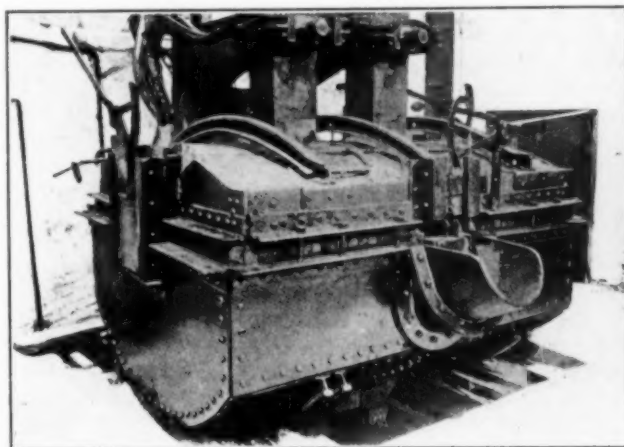


FIG. 1—VIEW OF GRÖNWALL STEEL FURNACE.

One of the latest types to be introduced—and one that possesses several distinct advantages over others—is the Grönwall furnace. This is the invention of three Swedish engineers—Messrs. Assar Grönwall, Axel Linblad and Otto Stalhane, of the Aktiebolaget Elektrometall of Ludvika, Sweden.

Having originally worked with various types of induction furnaces without great success, they decided to design an arc furnace which would embody the mechanical and electrical advantages of existing types, at the same time eliminating their weak points.

The invention consists in the method of supplying electric energy to the furnace, the form used being two-phase alternating current.

Furnace Construction.

As will be seen from the illustration (Fig. 1) the furnace has two carbon electrodes passing through the roof. These electrodes carry the two phases, the current arcing across from them to the charge, passing through this, and then through the basic lining to the neutral return, which is a carbon block fixed in the bottom of the furnace.

The iron tank of the furnace is lined with magnesite brick, and inside this lining the hearth of dolomite or magnesite is rammed. (See Figs. 2 and 3.)

The carbon block forming the lower terminal is set in a casting bolted to the furnace bottom, and good electrical con-

tact is secured by a special mixture which is used as cement. The top of this bottom electrode comes level with the brickwork so that it does not project into or in any way weaken the basic lining. The hearth of the furnace is not broken by any projections, being similar to that of an ordinary basic open-hearth furnace, repairing after each charge being effected in the same manner.

The roof consists of a channel iron framework holding silica bricks, and is detachable, so that when the bricks are worn thin it can be taken off and replaced by a new one, this

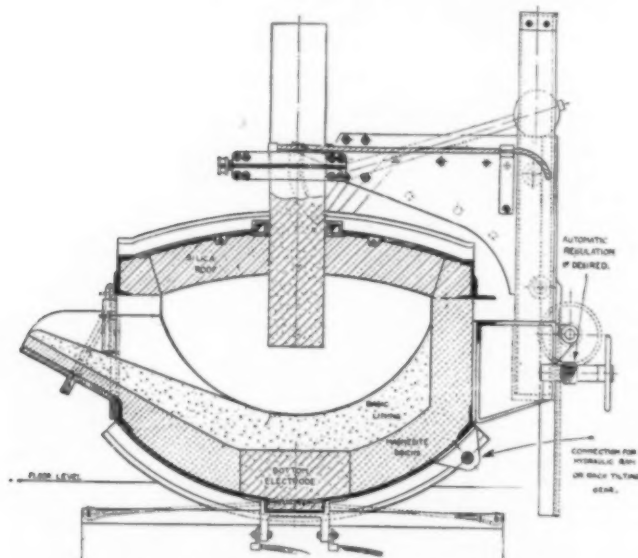


FIG. 2—SECTION OF GRÖNWALL STEEL FURNACE.

operation of changing only requiring about twenty minutes.

There is a working door at each end of the furnace and another one at the tapping spout. At the back of the furnace are the cranes for raising and lowering the electrodes. This is the only method of regulating the current and the electrode cranes can be operated either by hand or by automatic regulators.

The electrode holders for the electrodes passing through the roof offer three distinct advantages over the usual types employed:

1. The electrode holder at the same time grips the electrode and makes electrical contact.
2. The construction is simple and efficient; the holder consists of strong iron castings and T irons, a construction which will stand the rough usage to which it is exposed, and at the same time does away with complicated parts, such as springs, steel bands, etc., which are liable to get out of order.
3. The improvement is introduced of making the electrical contact low down on the electrode, thereby reducing to a minimum the loss of energy due to the resistance of the electrode.

The furnace is mounted on curved rails and is tilted by hand wheel gearing or by hydraulic cylinders. These rails rest on channel irons supported by strong foundations. The furnace construction, although simple, is cheap, strong and efficient.

Electrical Equipment.

The high-tension supply, if three-phase, is transformed by means of two single-phase oil-insulated self-cooling transformers, with Scott's connections to the tension suitable for the furnace, viz., 65 volts. Tappings are, however, provided on the primary side of the transformers for obtaining 50, 60, 70 or 80 volts.

The switch gear consists of an automatic oil switch, with cut-outs for overload, in circuit with the high-tension mains. On the low-tension side there is an ammeter on each phase and one on the neutral return. In addition a voltmeter with a three-way switch shows the voltage of each phase, and also

that between phases. The power consumed is measured by a watt-hour meter.

The current is conveyed from the transformers to the electrode holders by means of flexible bare copper cables, and from the bottom casting to the transformers by similar cables.

Advantages of the Two-Phase System.

Practically all power supplies are either two- or three-phase high-tension current. It is, therefore, a great advantage of this system that such current can be used direct by means of static transformers without the intervention of costly running plant.

Where the available supply is two-phase the furnace is coupled directly to the mains through two single-phase static transformers, giving suitable voltage in the furnace. In the case of a three-phase supply, as mentioned above, the current is transformed into two-phase by means of Scott's system of connections. In this manner a great saving in first cost and a high economy in working are gained as compared with the rotary transformers (converters or motor-generators) required for converting three-phase current into single-phase current.

In the Grönwall furnace each phase is connected to one of the top electrodes. Each arc is, therefore, formed independently of the other, so that if one arc is broken the other remains, while in furnaces with arcs in series the breaking of one arc also breaks those remaining. The Grönwall system ensures steadier running. Assuming the furnace to take 500 kw, i. e., 250 kw per phase, if one arc breaks only about 250 kw are immediately switched off the mains, whereas with the arcs in series the whole 500 kw would be cut off, causing a much bigger fluctuation in the power supply. This feature is so marked that in Sheffield, England, where one of these furnaces is installed, the authorities at the municipal power

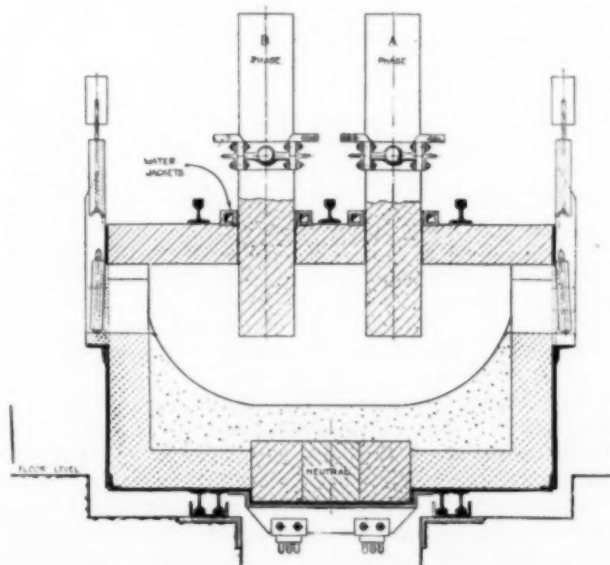


FIG. 3—SECTION OF GRÖNWALL STEEL FURNACE.

station permit the furnace to be started on a cold charge during their peak load.

In all arc furnaces the arc is, of course, the heat generator. During the melting stage the current passes from the electrodes to the charge, and as small pieces of this melt from under the electrodes from time to time the arcs are being continually made and broken. With a single-phase furnace with the arcs in series, this means a continual switching on and off of the entire current, with consequent disturbance of the supply system, loss of time in melting and loss of heat in the furnace by reason of the discontinuity of the supply of energy.

Apart from electrical considerations, it has been found that this two-phase system has decided metallurgical advantages.

The passage of the current through the basic lining uses very little energy, but it raises the temperature of the bottom sufficiently to prevent any metal cooling and solidifying in the bottom of the bath. The lining is not damaged in any way by the passage of the current, and experience has proved that there is no appreciable increase of wear and tear on the bottom.

The arrangement of two arcs in parallel with a neutral return through the bottom produces a vertical as well as a horizontal circulation of the metal in the bath. This facilitates chemical reactions by bringing all parts of the charge successively into intimate contact with the refining slag.

Due to this circulation, heat is taken away from the very hot region below the electrodes, so that the reflection of heat from this part on to the roof is much less than with single or three-phase surface heating, the result being to prolong the life of the roof. The roof of these arc furnaces is made of silica brick, so that when unduly overheated pieces of silica drop down into the basic slag the equilibrium of the reactions would be upset. The longer the roof can be made to last the greater is the reduction in cost and the less silica will pass into the slag per charge.

The process of steel-making in the Grönwall furnace is the same as that employed in the earlier and better known types of furnace. The quality of the resulting steel is so well known that it needs no emphasizing. It is generally admitted, however, that the absence of oxygen from the steel partly accounts for its remarkable properties. The process of deoxidation carried out in the Grönwall furnace is facilitated by the circulation of the charge referred to above.

The advantages of the Grönwall furnace may be summarized as follows: The utilization of two-phase current from a three-phase supply eliminates rotary transformers; an even distribution of heat is obtained throughout the charge, and at the same time a circulation of the molten metal is set up, resulting in improved refining conditions.

Toronto, Ontario.

The Remelting of Ferromanganese and the Deoxidation with Molten Ferromanganese.

In *Stahl und Eisen* of Aug. 24, 1911, appeared an article by J. BRONN and W. SCHEMMANN, of Rombach, Lorraine, giving some valuable information about the use of molten ferromanganese.

In view of articles recently published about this subject (*I. Tr. Rev.*, 1911, March 16, page 526) the authors deem it necessary to give some practical advice. They fear that on account of the difficulties and failures experienced in remelting ferromanganese at the present time in cupola furnaces, open-hearths or electric furnaces, steel men may lose confidence in this process and abandon it, although if properly carried out it has very important advantages.

In 1906 the Rombach steel works began to remelt ferromanganese. The molten metal was poured into the ladle, together with the heat from the converter, thus preventing any waste of ferromanganese due to smaller pieces remaining in the slag. Any reduction of phosphorus from the slag back into the bath, which hardly could be prevented before, was avoided.

Samples analyzed from all the ingots gave not the least differences between the first and the last ingots. This proves a quick and uniform distribution of ferromanganese in the bath.

The casting into ingots proceeded very smoothly and in none of the 1000 heats with molten ferromanganese any difficulties were experienced in casting. Even "dead heats" could be poured.

It is a surprising fact that heats with such a low manganese content as 0.22 per cent to 0.25 per cent could be rolled easily and with only a small percentage of "bad ends."

This permitted a considerable saving of ferromanganese per ton of steel. Further small pieces of ferromanganese, such as result from long storage, could now be used without loss, while in former practice a good deal of the smaller pieces were always lost with the slag.

These favorable results induced the Rombach Steel Works to adopt the process on a larger scale. There were some difficulties involved in the problem how to melt the ferromanganese. Cupola and open-hearth furnaces could not be considered, since too much of the manganese would be oxidized or lost with the slag.

On account of the low boiling point of manganese (1900 deg. C., against that of iron, 2450 deg. C.) the ordinary electric arc furnace is also unsuitable. In a test made with a small electric furnace such an immense smoke of oxide of manganese was developed (the manganese evaporating first as such, but being at once changed into oxide) that the laborers were unable to operate the furnace and the trial was abandoned after half an hour.

Another difficulty was found in the method of feeding electrical energy to the furnace. Girod's bottom electrode did not work at all. After many trials in an electric furnace of 300 kw to 400 kw it was found that the bottom electrodes were exceedingly corroded and penetrated by the ferromanganese. Nearly a dozen different types and sizes of bottom electrodes were tried out, but none lasted more than three days.

Investigation of these steel-bottom electrodes showed a transformation into a high-percentage and very brittle ferromanganese. Observations made by Wüst and Felsler have also proven that iron in contact with highly heated ferromanganese absorbs a great amount of carbon and especially of manganese. The use of the bottom iron electrodes was therefore discontinued.

Another problem had to be solved in the construction of the hearth of the furnace so as to prevent leakage. Since only small portions of the bath were taken from the furnace they had to be heated to a high temperature in order to prevent any freezing during transportation. But this hot and very liquid alloy in the furnace penetrates easily into the porous lining of the hearth and causes undesirable damage. These troubles were overcome by a special lining.

Numerous tests were made until a final satisfactory solution was found by using low-tension electric currents. In opposition to the ordinary arc furnaces with a drop of 45 volts to 75 volts between electrode and bath, a "far lower" tension was employed. (The exact voltage is not given in the article.)

The melting process then occurs smoothly without any of the troubles mentioned before. For example, the furnace contained more than 1.5 tons ferromanganese and after six days of continuous operation the furnace content showed not the slightest increase in percentage of iron or decrease in manganese.

With such low-tension current the distance between electrodes and bath surface could be reduced to a minimum without increasing unduly the consumption of electrodes (which was only 1 cm, or 0.4 in., per working hour as a weekly average). But the consumption of electrodes may be still reduced considerably.

It seems that in most cases the saving of manganese, which amounts to 35 per cent of the usual consumption of manganese, pays several times for the costs for remelting.

The results already obtained make it possible, according to the authors, to produce in one heat at reduced cost from Thomas iron an excellent quality of steel which may in many cases replace the ordinary basic open-hearth steel.

The Röschling Iron & Steel Works in Völklingen have also introduced this process in their mills. The remelting was undertaken in a small Röschling-Rodenhauser induction furnace, the results being satisfactory.

R. J. W.

A Simple Method for the Determination of Methane in Producer Gas and Enriched Water Gas.

By S. H. WORRELL.

The constantly increasing use of producer gas and enriched water gas for power in gas engines suggests the necessity for either a daily analysis or a b.t.u. determination with a calorimeter. The analysis is necessary to determine, oftentimes, just where the trouble lies in the process of making the gas. The calorimeter is useful in determining the value of the gas when bought on a contract of a rated b.t.u. delivery.

If an analysis is to be made, the simpler the method used the better. The following method is in almost daily use in the University of Texas Testing Laboratory. The apparatus used is a modified form of that put out by the United Gas Improvement Company. Instead of the graduated burette with a bulb and three-way stop cock, a burette calibrated its entire length, without a bulb, and having a two-way stop cock is used with a water jacket to maintain a constant temperature.

One way of the cock is attached permanently, that is, it does not have to be removed during the analysis, to a potassium hydroxide pipette. This leaves the other branch open to attach the various other bulbs in succession during the analysis.

The method in detail is as follows. One hundred cubic centimeters of the gas at atmospheric pressure are taken. Then as each constituent is absorbed the burette is read, the amount so obtained is subtracted from the previous reading and this difference is at once the percentage of the constituent under consideration. This subtraction is all the arithmetical calculation necessary in this method, the final residue being nitrogen.

The carbon dioxide is first removed with potassium hydroxide solution contained in the permanent pipette, then the "illuminants" are removed with bromine solution, the gas being run back into the KOH pipette to remove the bromine before the reading is made. It is then run into the phosphorus pipette to remove the oxygen. The fumes of phosphorus pentoxide thus formed are soon absorbed by the water in the phosphorus bulb. The time consumed for these three constituents is from ten to fifteen minutes. A three-minute sand glass will be found to be quite a convenience in this connection. The carbon monoxide is then removed with cuprous chloride in acid solution. This particular constituent requires that the pipette be disconnected from the burette and thoroughly shaken for from two to three minutes to produce complete absorption. The gas is then transferred to the other side of the pipette, a double pipette being used, and shaken for another three minutes. I have never found that the second treatment was necessary but always make it as a precaution. The gas is then run into the KOH bulb again to remove HCl fumes, after which it is measured again, of course. It is next passed through a palladium oxide tube contained in a beaker of water heated to about 90° C. to 100° C. It is best to pass the gas slowly at least three times, renewing the hot water or reheating it as may be necessary. It usually takes about fifteen minutes to make this absorption. The gas is then brought back to room temperature by replacing the hot water with tap water and recirculating the gas, after which it is measured. To regenerate the palladium oxide tube simply attach to the filter pump and pull air through it as long as may be necessary; I usually let it run for about thirty minutes.

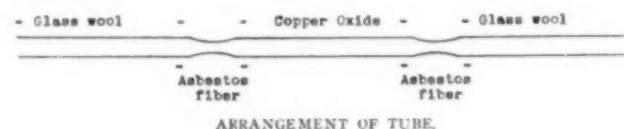
The gas, having been cooled, is measured and then run into the KOH pipette. If this treatment causes loss in volume it shows that not all the CO was absorbed by the CuCl solution and the percentage of CO is accordingly corrected by the diminution in volume here shown. Before the next analysis is made the CuCl solution is renewed.

There now remains methane and nitrogen. It is customary to run a measured amount of oxygen or air with a portion of

this residue and explode it in an explosion pipette. In either case an involved arithmetical calculation is necessary. One never knows how much of this residue is nitrogen and how much is methane, hence cannot tell how much air or oxygen to add. If too much is added the result may be a dilution beyond the point of a successful explosion. If not enough is added the result is equally unsatisfactory. If the correct amount is added the result may be too satisfactory. Even if the apparatus is not destroyed by a successful explosion, the analyst acquires the habit of throwing the switch with one hand and dodging behind the other.

If the methane is known to be low as it generally is in producer gas and frequently is in enriched water gas, the hydrogen must either be left in (and there is no check on the CO unabsorbed by CuCl) or a measured quantity of hydrogen has to be added. This involves further arithmetical calculation with results equally as unsatisfactory as above.

To obviate all this the idea was conceived of burning the methane to carbon dioxide and water by passing it through red-hot copper oxide. A fairly thorough search through the chemical journals failed to reveal anything published on the subject. It was then decided to try it, which was done in the following manner. The tip of a hard-glass arsenic tube was cut off, thus giving a straight tube shaped as in the following illustration:



The first section is loosely filled to the constriction with glass wool to take up the space and so restrict the volume. The constricted portion is then filled with asbestos fiber. The central section is then filled with wire oxide of copper such as is ordinarily used in organic combustion analysis. It should be ground in a mortar to about forty mesh and the fine dust screened out with an 80-mesh or 100-mesh screen. This allows free passage of the gas but fills the tube thoroughly. The outside of the tube is then wrapped around with about two layers of copper gauze or screen. This prevents the blaze striking directly on the glass and at the same time gives a uniform heating. The gas is then passed slowly through this heated copper oxide at least twice. When it cools to room temperature it should be measured. Any diminution in volume other than that due to the oxygen of the air enclosed in the tube when the combustion was begun is due to hydrogen not removed by the palladium oxide. The gas is then run into the KOH pipette to absorb the carbon dioxide formed. The loss in volume is the methane and is read direct in percentage, since one volume of methane on burning gives the same volume of carbon dioxide. There is, of course, an error introduced by the fact that the tube contains air and the oxygen in this is consumed, thus diminishing the volume by a certain amount. This amount can be determined by filling the apparatus with water, allowing the water to drain out and measure the volume so obtained. This total volume will hardly exceed 1 c.c., and the oxygen content in this cubic centimeter is the correction to be applied if desired.

The copper can be reoxidized by passing air through the red-hot tube. This method has been found applicable to percentages of methane as high as ten. If found necessary a second tube can be used or the same one reoxidized while still hot and the process repeated.

Testing Laboratory.

Bureau of Economic Geology and Technology.

University of Texas, Austin, Tex.

The Elmore flotation process at Sulitelma Mines produced in the month of April, 1911, a total of 3924 tons of concentrate containing 6.9 per cent copper.

Sizing with or Without Screens.

By CARL F. DIETZ AND DYKE V. KEEDY.

The greater interest taken in recent years in the possibilities of the dry concentration of ores has brought out, aside from marked improvements in magnetic and electrostatic processes, several new and ingenious forms of dry concentrators, or rather tables, which have successfully overcome the difficulties of the early attempts at dry concentration. The prejudices against dry concentration in general, which in some instances may be justified by the many failures of the past, are most generally the result of ignorance or a disregard of the fundamental principles involved.

It is not the purpose of this paper to discuss the relative merits of dry vs. wet concentration. It may, however, be mentioned that each has its own field. Only a careful study of the mixture to be separated, together with local conditions, can determine which system is the more desirable. Such decision should, of course, in every case be verified by the treatment of a representative quantity of the material. While it is evident that close and accurate sizing as a requisite to successful dry separation, especially in the case of a complex mixture showing close specific gravity differences, is gradually becoming better understood, it remained for the advent of the Keedy ore sizer to make the application of a system of accurate and close sizing of practical importance. Reference may be had to *Engineering and Mining Journal*, Feb. 5, 1910, "The Keedy Sizer for Classifying Complex Ores."

This machine is based on diametric sizing, that is, the grades are actually measured by the aperture of the screens, limiting each grade, and therefore the ratio in volumes between the largest and smallest particles in any one grade is as the cube of the apertures of the limiting screens. This law, while it is applicable to all minerals breaking into massive particles, is directly affected by the coefficient of fracture. The latter is variable, and as yet extensive investigations have not resulted in determining a suitable factor for each of the usual minerals. Hopes, however, are entertained that something of value in this connection may soon be developed and published. It has been found in practice that the law of cubes gives very satisfactory results and can be relied upon with the exception of certain minerals, such as mica and specular hematite, which break into flat scaly particles, presenting a large area as compared to their volume.

Exception was taken to the cost of sizing an ore into, say, 12 or 15 grades because of the complexity of the apparatus and flow sheet, wear and tear and cost of screen renewals. When considering the old methods of using shaking or bumping screens or reels, the objections are well founded, but are no longer tenable with the more modern form of apparatus. The system is compact, noiseless, dustless and requires but small power. Attention required is at a minimum and screen renewal cost, while varying with the class of material treated, is well within the limits of expense which any given ore can stand if it is worth treating at all.

The objections to the cost of grading in this manner, even though actual operations in a number of instances prove them to be unfounded, still continue and have caused the minds of metallurgists and inventors to seek other means.

Recently there have been given to the mining and metallurgical world some details of a new and ingenious device designed to size ore into any number of grades from 10 to 30 without the use of screens. Details of this new machine, invented by Messrs. C. L. McKesson & B. P. Rice, of Colorado Springs, are presented in *Mining Science* of Sept. 14, 1911, and in *METALLURGICAL AND CHEMICAL ENGINEERING* of October, 1911.

This new "screenless sizer," in the operation of which the laws of gravity are reported as having but a very slight influence, is capable of some rather remarkable performances, and the published results of an official test will bear some close mathematical analysis. It is barely possible that this machine

performs a function as yet apparently not recognized by the inventors or their associates, and if such is true, the apparatus should have a wider range of usefulness than would be accorded to a machine intended to be used as only a sizer.

The measurement of efficiency of any sizing operation is determined by test screening any particular grade through screens which supposedly mark the limits of that particular grade.

In preparing a complex ore for dry table separation it is necessary to make as many sizes as may be demanded by difference in specific gravity between the minerals which vary the least, so that the most perfect stratification may result and that clean zones of minerals, with a minimum of accidental middlings, may arrange themselves on the tables; these conditions can result only when the ratio of volumes between the largest and smallest particles, in any one grade, equals or is less than the ratio between the specific gravity of the two closest minerals.

As an example, in the case of iron pyrite and zinc blende, in which the respective specific gravities are 5 and 4, let V_p be the volume of the pyrite particle, while V_b is the volume of the blende particle. Then

$$\frac{V_p}{V_b} = \frac{5}{4} = 1.25$$

As pointed out, grades in which such ratios exist can be produced by considering the apertures of the screens, and if S_1 is the aperture of the coarse screen while S_2 is the aperture of the fine screen, we have

$$1.25 = \frac{S_1^3}{S_2^3}$$

Of course, a series of screens from, say, 20 mesh to 200 mesh, which throughout would have its aperture cubes bear a relation to each other as 1.25 to 1, leads too far for one sizer and necessitates two machines, as some 22 sizes would be required.

It may be mentioned that in the case of a very complex Mexican ore such a series was employed with the most gratifying results, showing sharp lines of demarcation between the fields occupied by galena, pyrite and blende, so that it was not necessary to cut a zinc-iron middlings.

This, of course, is an ideal condition, but it was found that the system could be much simplified by making only 13 sizes and collecting the accidental pyrite-zinc and zinc-gangue middlings, which, by resizing together over a machine dressed differentially, that is, making between-sizes, prepared them for successful treatment for the reason that the particles were rearranged, and therefore they could not all again report themselves as middlings.

The McKesson screenless sizer is designed to make as many as 30 grades in one operation, which, if the ratios of volumes between the largest and smallest particles in any one grade should prove to be as above shown, would be admirably suited as feed for a dry concentrator.

The aperture dimensions of the screens used to test the various grades must be known, but for illustration we may arbitrarily use the following in discussing the results of the official test on the McKesson sizer.

Mesh.	Aperture inches.
24	.0342
28	.0282
30	.0268
34	.0229
40	.0185
44	.0172
50	.0145
66	.0102
82	.0079
97	.0058
116	.0047
139	.0033
172	.0028

The results on p. 587 show that, except in the two coarse grades, efficiency is based on the material reporting in the two heaviest screen products, which, when taken together, show a large volume-ratio between the largest and smallest particles lying between the limiting screens used in determining measure of efficiency.

The results of a test on the McKesson machine making 15 products are reported to be as follows:

Product No.	Screen.	Per Cent.	Efficiency.
1.	— 24 + 30 + 40 — 40 15.60 81.20	81.20
2.	— 28 + 40 — 40 93.75 6.25	93.75
3.	— 30 + 34 + 40 + 44 — 44 10.92 50.00 32.83 6.25	82.83
4.	— 34 + 40 + 44 + 50 — 50 28.13 37.50 25.00 9.37	62.50
5.	— 40 + 44 + 50 + 66 — 66 7.28 25.00 64.06 3.12	89.06
6.	— 44 + 50 + 66 + 82 — 82 6.25 68.75 21.87 3.12	90.52
7.	— 50 + 66 + 82 — 82 37.50 53.13 9.37	90.63
8.	— 50 + 66 + 82 + 97 — 97 9.37 62.50 25.00 3.13	87.50
9.	— 50 + 66 + 82 + 97 — 97 6.25 50.00 37.50 6.25	87.50
10.	— 66 + 82 + 97 — 97 37.50 46.87 15.63	84.37
11.	— 66 + 82 + 97 + 116 — 116 25.00 50.00 21.87 3.13	75.00
12.	— 66 + 82 + 97 + 116 — 116 15.63 43.75 34.37 6.25	78.12
13.	— 66 + 82 + 97 + 116 — 116 6.25 37.50 43.75 12.50	81.25
14.	— 82 + 97 + 116 + 129 — 139 25.00 43.75 28.12 3.13	71.87
15.	— 82 + 97 + 116 + 139 + 172 — 172 6.25 18.75 43.75 25.00 6.25	68.75
Average.....			81.32

The efficiency, determined even in this most favorable manner, cannot be said to be high, and when considering the vitiating effect of the relatively large amount of "through" product recorded, and its detrimental effect on table work, the results cannot even remotely be compared to those obtainable by diametric sizing, when making even a lesser number of products and covering a wider range.

To quote from METALLURGICAL AND CHEMICAL ENGINEERING, page 554: "It will be noticed that the grading efficiency in some of the products is lower than in others, but it is claimed that this can be remedied by adjusting the cutters to throw the oversize and undersize into the preceding and succeeding grades respectively, and thereby increase the efficiency of the sizing in all grades."

In the above statement the absence of such positive action as exists when sizing diametrically is clearly admitted.

That the efficiency shown in one grade may be improved by throwing the over and undersizes into the preceding and succeeding grades respectively is obvious, but on the basis of the published screen analyses of the products, instead of thus increasing the efficiency of all the grades, those into which the oversize and undersize are cut would be more likely to suffer, with the result that, while one grade would be improved at the expense of its tonnage, the two grades affected by the improvement of the one would probably show not only a lower efficiency, but would represent a correspondingly larger proportion of the whole. If the action of the machine were positive the explanation of the possibility of improving the grades would have some force which, in view of the results, is lacking.

Taking the basis of the efficiency calculation as given, and applying the law of cubes, we can calculate the ratios existing between the largest and smallest particle in any one product, giving the machine the advantage of using the screens specified, which limit the efficiency calculation. We have then:

Product No.	Limiting Screens.	Per Cent. Efficiency.	Vol. Ratio of Largest to Smallest Particle.
1	— 30 + 40	81.20	3.04:1
2	— 28 + 40	93.75	3.50:1
3	— 34 + 44	82.83	2.30:1
4	— 40 + 50	62.50	2.70:1
5	— 44 + 66	89.06	4.80:1
6	— 50 + 82	90.52	6.20:1
7	— 50 + 82	90.63	6.20:1
8	— 66 + 97	87.50	5.40:1
9	— 66 + 97	87.50	5.40:1
10	— 66 + 97	84.37	5.40:1
11	— 66 + 97	75.00	5.40:1
12	— 82 + 116	78.12	4.70:1
13	— 82 + 116	81.25	4.70:1
14	— 97 + 139	71.87	5.40:1
15	— 116 + 172	68.75	4.80:1

Average ratio..... 4.66:1

In the practical application of close and accurate screening we have adopted, as a standard, a series of 13 screens between 20 and 200 mesh conforming to a harmonic series, thus producing 14 products, including oversize and fines. The average volumetric ratio is 2 to 1, while the efficiency, when measured by the screen aperture actually limiting the grade, averages better than 90 per cent, the only drop under 90 per cent being in the — 150 + 200 grade, which, with the screen area usually exposed, varies between 80 and 85 per cent.

The percentage of oversize is practically nil and the particles passing through the fine limiting screen, in testing the grade, are usually of such a size as to just pass through the aperture, but much too coarse to go through the next screen of the series. If the efficiencies in testing the products from the Keedy sizer were figured in the same manner as those for the McKesson sizer products the former would show an average efficiency of from 95 to 98 per cent, as against 81 per cent for the latter.

The series as developed and used by us as a standard is the following, in which screen cloths have been selected of apertures conforming with the law of cubes:

Mesh.	Apertures, Inches.	Ratio of Volumes of Largest to Smallest Particle.
20	.0410	1.722 to 1
24	.0342	1.722 to 1
28	.0282	1.783 to 1
34	.0229	1.867 to 1
42	.0183	1.959 to 1
50	.0145	2.012 to 1
62	.0116	1.953 to 1
74	.0089	2.214 to 1
86	.0068	2.242 to 1
109	.0054	1.996 to 1
125	.0041	2.284 to 1
150	.0032	2.103 to 1
200	.0026	1.865 to 1

Average 2.000 to 1

A comparison between these ratios and those of the screenless sizer work, entirely aside from the considerably better efficiency of diametric sizing, clearly indicates which products may be expected to yield the better table results.

The results of the McKesson test show that in a general way the efficiency varies inversely as the ratio of maximum to minimum particles in the grades, excepting in the extreme coarse and fines, but the most important characteristic revealed is the gradual gradation by size almost identical in its character with, but somewhat more emphasized than, that resulting from feed-

ing an unsized pulp over a dry table of the Sutton, Steele & Steele type.

It may be especially pointed out that in every instance in the screen analyses the coarse limiting screen used is of larger aperture dimension than the fine limiting screen of the preceding grade. The efficiency of products 8, 9, 10 and 11 is measured by exactly the same screens, the only difference in the grades being that they grow finer by reason of the fact that the oversize is gradually reduced while the undersize increases. The latter is also true of products 12 and 13.

the result would have been quite as perfect as that of the McKesson machine.

Mesh.	Feed.	Products			
		B	C	D	E
— 50					
+ 58	1.50	4.1	.10	.25	0
+ 68	.50	2.3	.45	.25	0
+ 90	9.50	16.0	10.00	7.25	2.50
+ 100	12.00	28.7	13.00	13.75	3.50
+ 140	14.25	18.0	19.25	14.75	5.00
+ 163	21.50	10.1	16.25	11.25	6.00
+ 200	18.00	14.1	24.00	26.75	32.25
— 200	20.50	6.5	16.00	24.00	49.00

Another notable instance—in the treatment of a mixture of

galena, pyrite, zinc and fluorspar, it was invariably found that the finer galena traveled under the pyrite, the finer pyrite under the zinc, and the finer zinc under the fluorspar. In air, a particle of zinc need be only 25 per cent greater in volume than a particle of pyrite to weigh the same, while a particle of fluorite need be only 30 per cent greater in volume than a zinc particle to weigh the same.

Now these particles of equal weight are acted upon equally, and necessarily produce a middlings which when screened over a cloth of proper aperture results in a concentrated product either on or through,

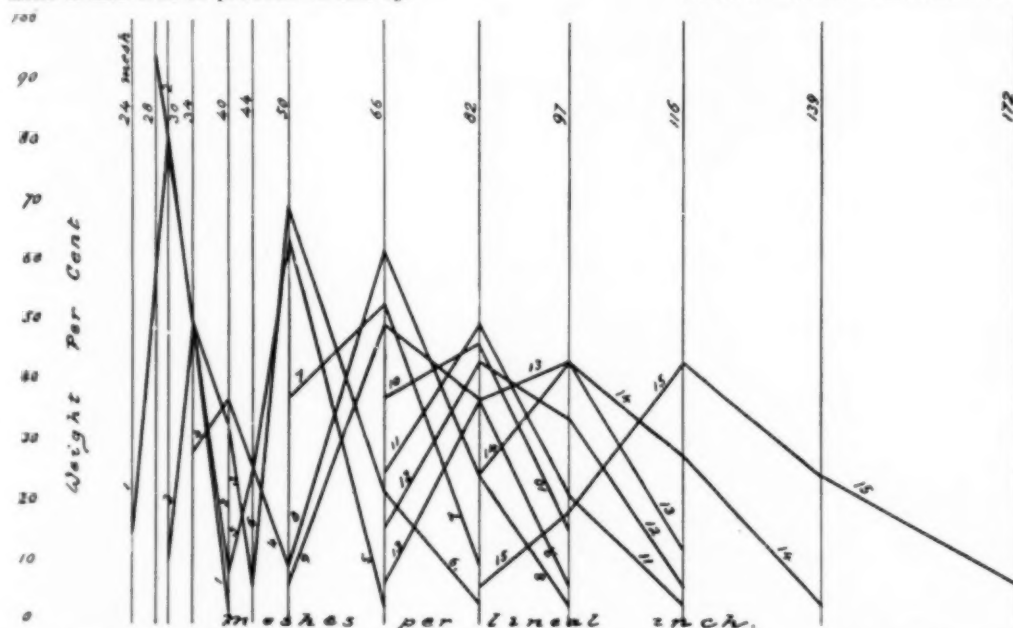


FIG. 1.—GRAPHICAL REPRESENTATION OF SCREEN ANALYSES OF PRODUCTS FROM A MC KESSON SCREENLESS SIZER.

This fact proves that the products very considerably overlap, and thus the sharp line of demarcation existing between diametrically sized grades is wholly lacking in the new method of sizing. The overlap of the products is graphically shown on Plate I, and it is to be clearly noted that no sharp line exists between any two or even three of the grades, while the overlap, in specific cases, is sufficiently great to completely destroy the effect of sizing. For example, products No. 7 and No. 15 are so poorly sized that —82 + 97 mesh material is contained in each.

With an average recorded efficiency of but 81 per cent, and that, too, on an average particle ratio of 4.66 to 1, the actual average particle ratio on the whole product exceeds 6 to 1, which is far too much to permit of anything like desirable table work.

The above discussion is, of course, all predicated on the statement that specific gravity plays a unimportant part in the operation of the McKesson apparatus.

That a machine of this general type will operate as a sizer has been pointed out, but the sizing is of an extremely indifferent quality, and specific gravity actually does play a part well worthy of notice. In one instance, an unclassified ore was treated on a Sutton, Steele & Steele table, primarily for the purpose of noting the sizing effect. The tendency was marked but the gradations along the periphery of the table were so gradual as to be characterized as only an abortive attempt at sizing. The feed to the table and the four products caught at the periphery gave the screen analyses on top of next column which are graphically represented in Plate II.

These results were obtained on a magnetic iron ore, and the products "b," "c," "d" and "e" clearly showed that the finer heavy magnetite particles had a distinct tendency to associate themselves with coarser gangue particles—the result of the ever-operating law of gravity. If 15 products had been cut,

or both, the heavier mineral reporting in the finer product.

The published screen analyses of the products from the McKesson sizer do not indicate that specific gravity plays no part in the operation, and the question obviously arises whether this machine does not perform a function of greater importance than the one which its inventors desire it to perform. Certain it is, on the basis of the recorded efficiencies and the evident gradual classification of the products, that the name screenless sizer is a misnomer. As far as the usual understanding of sizing is concerned, this machine is not a sizer, as its products overlap far too much. It would be more appropriate to call the apparatus a dry classifier, which in effect it is.

Mr. Charles J. Downey, in *Mining Science*, Sept. 14, 1911, page 247, states: "Concerning the theory of the McKesson method, I have no disposition to enter into a discussion of it, beyond what has already been said. An outline of the laws governing the selective action, in which differences of specific gravity play but a very small part (and then only in the case of widely different minerals, according to the engineers) will be the result of systematic and prolonged experiment. That the sizing is effected to a high degree of efficiency is apparent to an observer, while the operation of the machine has all the aspects of simplicity."

Now going back to the thought expressed above that this machine is essentially a dry classifier, all speculations vanish as to a new principle having been stumbled upon, and viewing the action as one of classification is the more tenable, when considering the analogy between the operation of the McKesson machine and the Sutton, Steele & Steele dry table.

It is difficult to conceive of one making separations between minerals of equal size, on the basis of their specific gravity differences, while the other collects together the various minerals according to their size, regardless of their relative specific gravities. These two thoughts are distinctly contradictory.

If the McKesson sizer acts in the nature of a classifier, which is exactly what a table does (only the new machine by means

ready acceptance of an apparent overthrow of fundamental principles that have served us long and satisfactorily.

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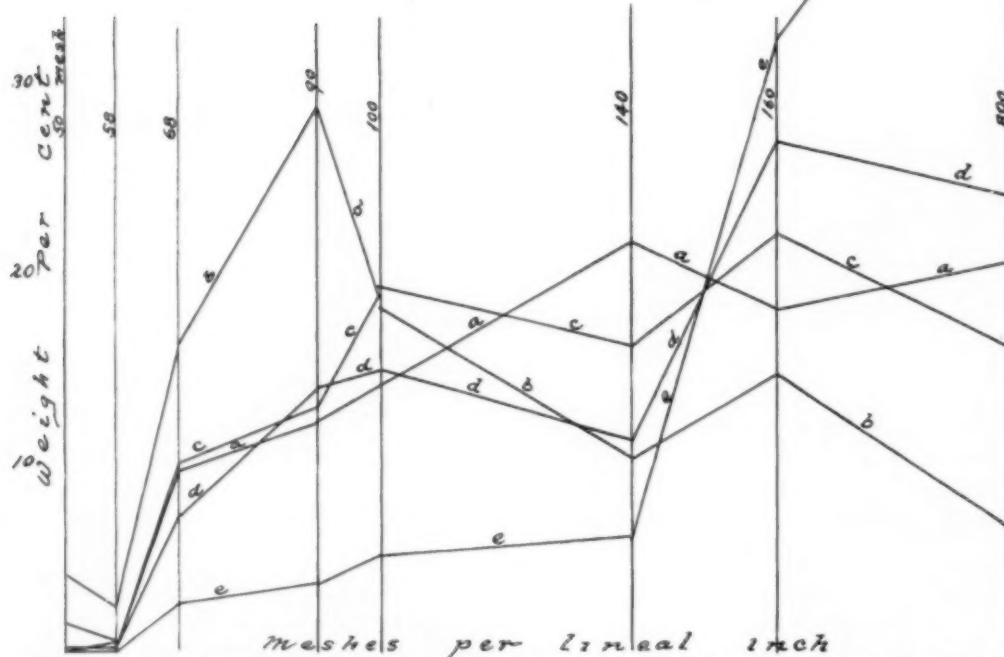


FIG. 2.—GRAPHIC REPRESENTATION OF SCREENLESS ANALYSES OF PRODUCTS FROM A SUTTON, STEELE & STEELE TABLE, WHEN FED WITH UNSIZED ORE.

The diagram on the top represents the Deck of Sutton, Steele & Steele Table, showing Peripheral Discharge of Products b, c, d, e.

of the deflectors and discharge arrangement takes advantage of and emphasizes the action), it must be shown by the character of the screen products of each grade.

A hydraulic classifier when making 15 products appears to deliver a fine string of perfect sizes, however a screen analysis and assays of the screen products reveal the true action, which is classification both by size and specific gravity.

In each grade it will be found that the lightest mineral reports, to the greatest extent, in the coarsest grade; while the heaviest mineral will be found, to the greatest extent, among the finer particles of the particular product under consideration.

This same action is to be expected in a machine of the McKesson type.

It can readily be verified by testing a mixed sulphide ore and screen—analyzing the products, just as was done in the reported official efficiency test.

The investigations should, however, be carried further, and each of the screen products should be assayed for the principal metals contained in the mixture, thus readily establishing the actual facts of the operation.

We anticipate that results of such an investigation will show each product to be made up of coarse gangue, finer zinc, somewhat finer pyrite, and still finer galena. If this explanation of the operation of the new machine is found to be true, we will no longer be confronted with a new and paradoxical theory to worry the research metallurgist, but will realize the ease with which appearances may deceive and the danger of a too

The development of a dry classifier should have a wider field of application than a sizer, if its capacity is sufficiently high, since, if the machine acts as herein discussed, its products should make an admirable feed for a wet table.

That the new machine is not in any sense a sizer of mixed minerals seems well established, and that it follows the laws of classification in which specific gravity plays an important part is strongly indicated by an analysis of the published results.

We may express a hope that the inventors will shortly supply the metallurgical world with further, more complete data, preferably along the lines suggested, including a screen analysis of the feed to the machine for purposes of comparison, so that speedily a more definite understanding of the principle involved in the performance of this ingenious new contrivance may be had.

Boston, Mass.

Producer gas is the source of power for seven mines in one of the oldest mining districts in Mexico, viz., in the district of Sultepec in the State of Mexico. Electric power has recently been brought into the district and is furnished at a cost of \$170 per hp-year (presumably Mexican currency.) The mine pays for the construction of the substation, and the power discounts 25 per cent of the cost from the power account until the line is paid for.

The Eclipse filter is an Australian device for filtering cyanide or other valuable solution from ore pulp. It consists of a wooden frame 28 ft. long, 4 ft. wide, and having a fall of 3 ft. in its length. The filtering medium is an endless three-ply rubber belt, $3\frac{1}{2}$ ft. wide, with numerous holes punched through it. Over the belt, and attached to it at the sides only, is a strip of filter cloth. In the middle of the machine is the filtering or squeezing gear, which consists of a series of cast iron rollers. The belt travels in a horizontal position except as it passes between the squeezing rollers, when it assumes a vertical position, being folded along its median line. The pulp is fed onto the belt at the upper end of the machine and as the belt passes between the rollers the solution is squeezed out of the pulp, the latter forming a cake which is discharged at the lower end of the machine. If the cake has to be washed it must be passed through a second machine. The capacity is about 900 lb. per revolution, or 650 tons daily. It requires about 3 hp. One is installed and working in the Associated mill, Kalgoorlie, and is more fully described in the August issue of the *Mining & Engineering Review* (Melbourne).

Manufacture of Steel in the Girod Electric Furnace.*

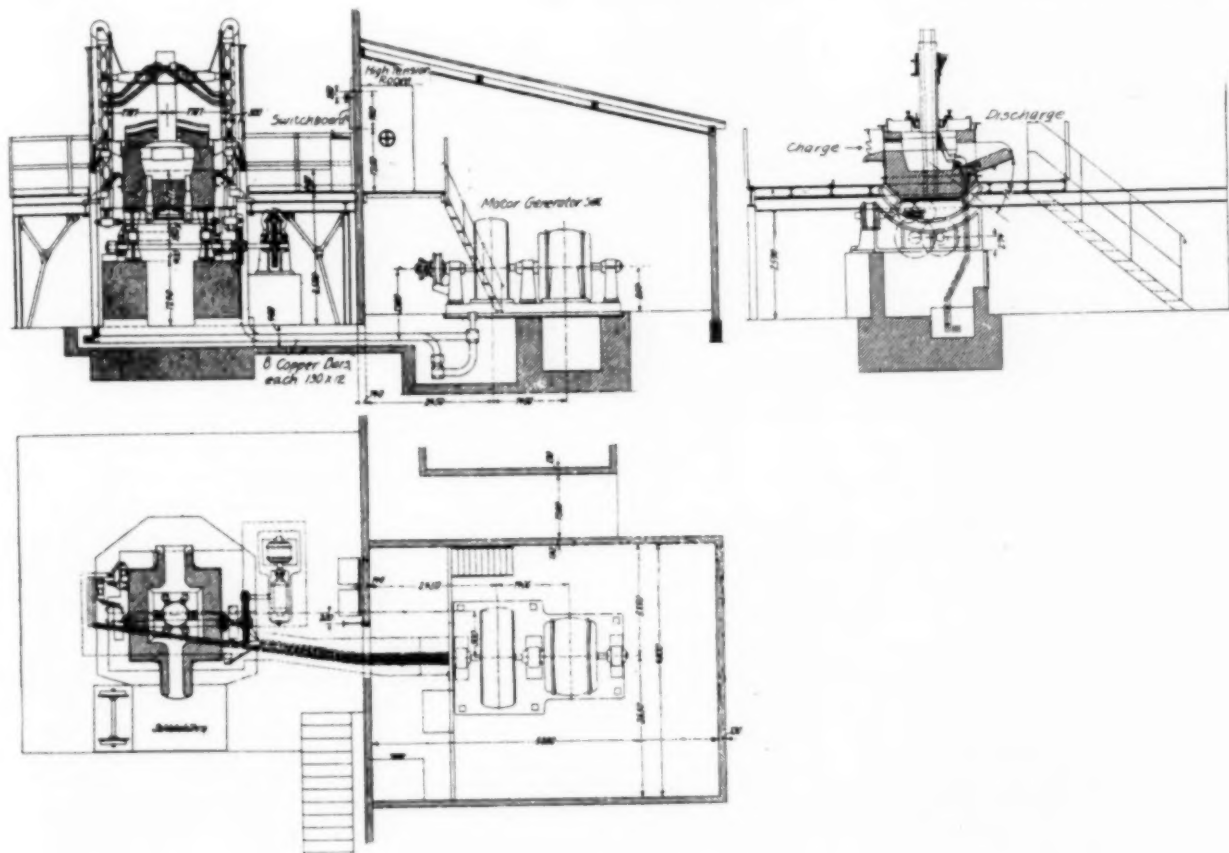
BY DR. A. MUELLER.

The Gutehoffnungshütte has installed a three-ton experimental Girod furnace in their Martin Works No. 1 for the purpose of electrically refining molten steel from their 25-ton open-hearth furnaces.

all of the cables are on the side of the furnace which faces the motor-generator.

In method II (Fig. 6) the cable to the carbon electrode is divided into two sections which run parallel to each other. The steel electrode is connected by the shortest path with the motor-generator set.

Method III (Fig. 7) is the one used since August, 1910, at the Gutehoffnungshütte. The current is conducted to the carbon electrode in a manner similar to method II. But while in



FIGS. 1, 2, 3.—ARRANGEMENT OF ELECTRIC FURNACE PLANT.

Figs. 1 to 3 show the equipment of the entire electric furnace plant, and the various details of the furnace and the electrical equipment. Fig. 4 is an outside view of the furnace. Only small details have been changed in the original design of Girod. The single-phase current for the furnace is furnished by a motor-generator consisting of a 3000-volt 95-amp. 575-hp three-phase induction motor (with a power factor of 0.92) and a single-phase 75-volt, 6700-ampere, 25-period, 500-kw generator (with a power factor of 0.8) with exciter unit.

These machines, as well as all conductors carrying high-tension currents are placed in a separate room adjoining the furnace room. The electrical conditions can be watched from the furnace by means of ammeters and voltmeters in the low-tension circuit and a kilowatt-hour meter in the high-tension circuit. The switchboard at the furnace contains the excitation-current regulator for the furnace voltage, the manual and automatic regulator for the electrodes, and the device for tilting the furnace.

The arrangement of the electrical-conductors is of great importance. There are three methods in use for conducting the current to the carbon electrode at the top and the steel electrode in the bottom, employed in the Girod furnace.

In method I (Fig. 5) the shortest path from the motor-generator set to the carbon and steel electrodes has been chosen;

methods I and II the bottom steel electrode is insulated from the furnace body itself (only the metal bath being in the

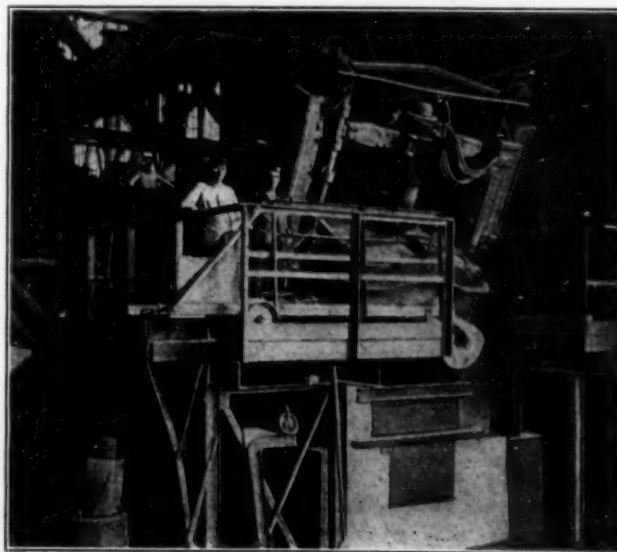
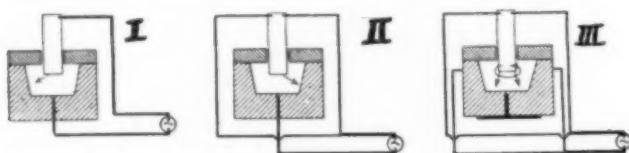


FIG. 4.—OUTSIDE VIEW OF GIROD FURNACE.

*Translated from *Stahl und Eisen*, July 20 and Aug. 3. Slightly abstracted.

electric circuit), the steel electrodes are here electrically connected with the furnace body. The arrangement of the conductors is symmetrical around the furnace.

With the first two methods the electric arc is deflected towards the side marked with the arrow, which results in a non-uniform heating of the steel bath and a rapid destruction of that side of the furnace which is exposed to the excessive direct heat of the arc. Method III avoids all this.



FIGS. 5, 6, 7.—ARRANGEMENTS OF ELECTRICAL CONNECTIONS TO THE FURNACE.

In view of the currents of high amperage and low voltage which with the Girod furnace is operated, there are considerable energy losses in the low-voltage electric conductors depending both on the resistance and the reactance. Method I, with its short conductors, was originally adopted to make these losses small. But the destruction of the furnace walls and roof on the side to which the arc was blown as indicated by the arrow was so excessive that it was cheaper to incur the

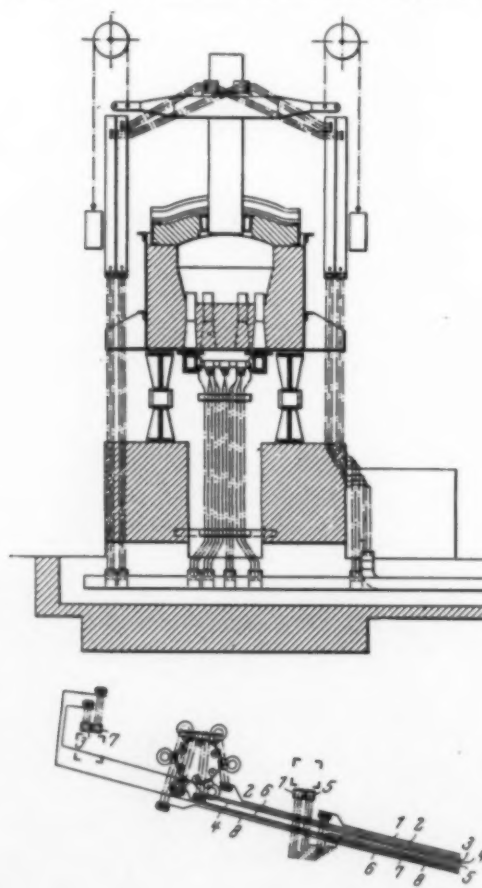


FIG. 8.—OLD ARRANGEMENT.

expense for greater lengths of cables and the losses in the same than the expense for the furnace repair.

It is good practice, therefore, to use as low a frequency of the alternating current as possible, to avoid a material of high permeability for the furnace construction and to try to keep the iron masses as far as possible out of the electric circuit.

The electric arc is forced to distribute its heat uniformly

over the bath by arranging the cables symmetrically around the furnace as in Fig. 7, method III. In this way the magnetic fluxes, formed by the iron shell, are reduced to a minimum. The magnetic field around the carbon electrode becomes uniform, the heating effect of the arc becomes uniform, and the current passes through the charge in a more regular order.

The advantages of this arrangement of the cables, from an electrical and metallurgical standpoint, are the following:

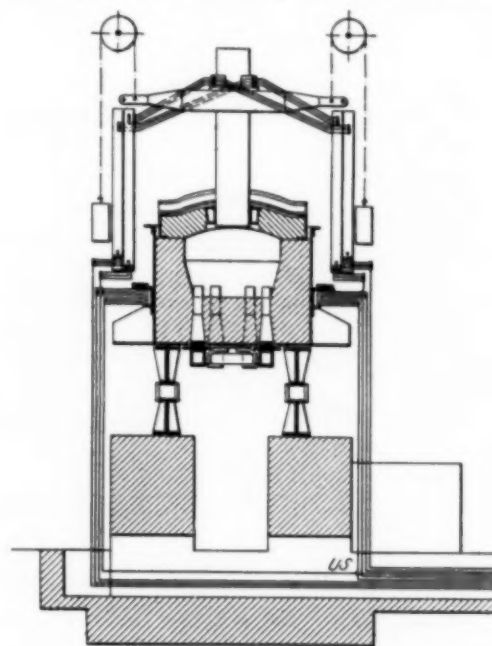


FIG. 9.—NEW ARRANGEMENT.

1. The electric arc circling about the periphery of the carbon electrode causes a strong agitation of slag and metal. This accelerates the speed of reaction between the slag and the iron bath and therefore reduces greatly the time of refining.

2. The arches of the roof as well as the furnace walls receive a more uniform radiation from the electric arc and last longer.

3. The saving in energy consumption is 10 per cent compared with the former arrangement.

4. Copper bus bars can be used instead of cables.

5. The metal bath is heated more uniformly throughout and the resulting product is more uniform.

6. Current interruptions due to the rupture of the arc are avoided, eliminating the resulting rushes of current on the motor-generator set and allowing an easier melting of cold charge than with method II.

7. The consumption of the carbon electrode is more uniform, while with the former arrangement one side of the electrode was consumed more quickly than the other, resulting in greater expense for electrodes.

Fig. 8 shows the older arrangement corresponding to Fig. 6, method II. The current passed in a 23-ft. long circuit through the copper bus bars, marked 8, to the furnace. One half of these bus bars was connected with the bottom electrodes by means of twelve cables, the other half was distributed on both sides of the furnace. With the other electrode connection was made from the conduit both by means of cables and by bus bars. Three cables led on the right side from bus bars 1 and 5, three on the left side of the furnace from bus bars 3 and 7 up to the carbon electrode, while from bus bars 2, 4, 6, 8, three cables each led to the steel electrodes which were electrically connected with each other but insulated from the furnace body.

In the new arrangement (Fig. 9) the six steel electrodes are electrically connected by means of a copper ring and plate with each other and with the furnace body. The entire current is

conducted close to the furnace by eight parallel copper bus bars, and divides at the current distribution point U S in such a way, that opposite currents pass alternately along both sides of the furnace to the pivoting point, where each bus bar is connected by three cables (more recently, by flexible copper bars) either with the furnace body or with the carbon electrode, as clearly shown in Fig. 9. This arrangement with the low voltage of operation overcomes all insulation difficulties and has reduced induction currents considerably. The rotating magnetic field around the electrode are uniform and very small. This arrangement also permits an easy construction and a saving of cables.

For the first 225 heats the furnace was operated only during the day and in many cases it was possible to tap the furnace five times in 12 hours under favorable conditions (See Fig. 13). In continuous operation it is possible, working day and night, to obtain 25 tons of steel from 8 tappings.

Since the open-hearth furnaces were only tapped every four hours, the electric furnace could not be operated continually. About 3 tons of the metal tapped from the open-hearth was poured into a heated ladle on a weighing scale, and then charged into the electric furnace.

But this method was found to be too expensive, since the temperature drop of the charge from the open-hearth to the electric furnace resulted in a rather high fuel and energy consumption. It was, therefore, finally decided to use a 23-ft. long launder which allows direct charging without any trouble. The weight of a charge can be adjusted with an accuracy of 200 kg. (400 lbs.) by the eye or by a watch.

The operation of the electric furnace is in the main like that of the basic open-hearth furnace, the metallurgical reactions being essentially the same, except the deoxidation. During the charging process, the current is 4000 to 6000 amperes; after the charging launder has been removed from the furnace by a crane, the amount of ore and lime required for the metallurgical reactions is shovelled into the furnace.

The combustion of the phosphorus is now started by a gradual increase in the temperature, while simultaneously the amount of carbon still present is reduced down to a minimum by oxidation. A portion of the sulphur is also removed during this period. This period of refining by oxidation is, of course, the longer the higher the carbon and phosphorus content of the charge.

When this period is over, the oxidation slag is removed by tilting the furnace and the last traces of the slag are carefully scraped off. Then petroleum coke or carbon electrode remnants are added for recarburising and a deoxidation slag is formed by adding lime, fluor-spar, and ferro-silicon. The deoxidation and desulphurisation of the bath now proceed and with proper operation a sulphur-content of 0.01 per cent can be guaranteed. After complete refining, the tap hole is opened, the current cut off, the entire furnace with the carbon electrodes tilted 40° from the horizontal, and the contents poured into a 4-ton ladle.

Lining.

The lining of the furnace must be constructed and maintained with great care. The arches are chiefly subjected to destruction. With the new arrangement of the conductors, which has resulted in greater uniformity of heat radiation from the arc and therefore greater uniformity of the wear and tear of the lining, the silica roof with attached cooling ring will stand 60 or 70 heats. The slightly curved furnace roof of silica brick reinforced by iron bands is insulated from the side walls of the furnace by a thin layer of asbestos and a new roof can be substituted within 20 minutes.

It should be mentioned that the contact areas of the silica roof with its magnesite brick edge resting on the basic walls of the furnace do not reach such a high temperature as to make these materials so-called electrical conductors of the second class, which would result in leakage of the current. More-

over, the carbon electrode usually does not come in contact with the roof-cooling ring so that the insulation is only used as an additional safeguard.

The hearth and walls were at first lined with magnesite, but dolomite is now used.

The ratio of the cross-section of the steel electrodes (Fig. 10) which are embedded in the bottom of the furnace chamber, to the rest of the bottom area is 1 to 16.

A repair of the bottom does not become necessary until it is required to repair the walls. The walls are attacked strongest at the slag line, but the damage can be repaired between charges. After 120 heats the walls must be re-lined. The hearth bottom must also be repaired then and filled up again to the original level, since it has meantime become about 2 in. deeper at the center (where the temperature is highest) than at the periphery.

The cost of an entire dolomite lining is about marks 350 (\$87.50), that of an entire magnesite lining marks 700 (\$175). Dolomite has proved so efficient that it is now used exclusively. The total maintenance cost for the lining per ton of steel is about marks 1.20 (30 cents.)

Heat Losses Due to Cooling.

Hearth-electrode cooling is still little understood and has even been called a great drawback to the efficient operation of

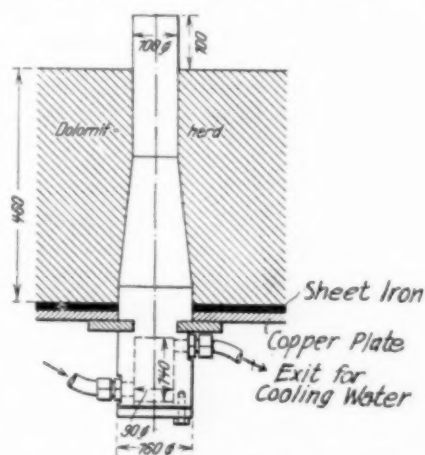


FIG. 10.—WATER-COOLED STEEL ELECTRODE.

the Girod furnace. Fig. 10 shows that only that portion of the soft steel bars which projects below the furnace body, is cooled by water, the bars being connected by a common pipe line.

The difficulties of maintenance of the lining of the Girod hearth are also often exaggerated. The dolomite hearth rammed by compressed-air rammers has lasted through more than 1000 charges and has never given any trouble in spite of different lengths of heats and very different compositions of the charges.

The estimate of the heat losses in the cooling water is often too high, so that the following calorimetric figures should be of interest. The hearth poles were cooled by water running at the rate of 4.9 liters per minute (1.2 gallons per minute), using 637 liters (168 gallons) for a heat of 130 minutes.

The temperature of the water flowing to the bottom electrode was 8° C, that of the water leaving the electrode was 21° C in the first 85 minutes and 23° C in the following 45 minutes.

The heat consumption, due to the cooling of the bottom electrodes was, therefore, $4.9 \times 85 \times (21 - 8) = 5414$ calories (kg), plus $4.9 \times 45 \times (23 - 8) = 3307$ calories (kg), hence the total loss of heat 8721 calories (kg), equivalent to an energy consumption of 10.1 kw hours or 13.7 horse-power hours.

Since according to Joule's law the total energy consumption is equal to $C = 0.2386 E J t$ calories, we have in this case with a tension of $E = 55$ volts and a current $J = 4800$ amperes during the total time $t = 7800$ seconds an energy consumption $C =$

$0.2386 \times 55 \times 8400 \times 7800 = 859,819$ calories = 1004 kw hours (measured by the meter) = 1364 hp hours. Hence the loss of heat due to cooling the bottom electrodes is 1.01 per cent of the total energy consumption and when producing 3500 kg (7700 lb) of steel, about 2.9 kw hours or 4 hp hours are consumed per ton of steel for cooling the bottom electrodes.

Let us now calculate the heat loss due to cooling the carbon electrode and the arched roof. 0.2866 liter per second (or 4.5

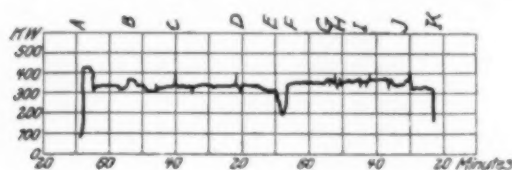


FIG. 11.—POWER FLUCTUATIONS DURING A HEAT.

(A tapping, B, C, D additions, E carburization, F slagging, G slag readily fluid, H bath boiling, I addition of ore, J slag not fluid, K charging.)

gallons per minute) pass through the cooling ring for the roof, or 2236 liters (590 gallons) for a heat of 130 minutes.

The temperature of the water entering the cooling ring was 9°C , that leaving it was in the first 70 minutes 21°C , and in the following 60 minutes 25.5°C .

This gives a total loss of heat of

$$\frac{2236 \times 70 \times 12}{130} + \frac{2236 \times 60 \times 16.5}{130} = 31.47 \text{ calories, equal to } 36.7$$

kw hours or 49.1 hp hours, so that the loss of heat due to the cooling of the carbon electrode is 3.65 per cent of the total

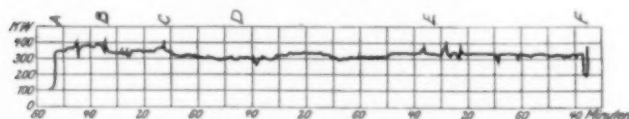


FIG. 12.—FLUCTUATIONS OF CURRENT WITH A HEAT IN THE FURNACE AFTER HAVING BEEN COOLED FOR 12 HOURS.

(F charging, E decarburization, D slagging, A, B, C additions.)

energy supply, 3500 kg. (7700 lbs.) of steel were produced so that 10.5 kw hours or 14.2 hp hours were lost per ton of steel for cooling the carbon electrode.

The Girod furnace with one carbon electrode requires about 0.65 cubic meters of water per ton of steel for cooling the carbon electrode, which is equivalent to 9000 calories. To cool the steel electrodes in the bottom the furnace needs 0.20 cubic meters of water per ton of steel, which is equivalent to 2500 calories. This heat loss is insignificant when compared

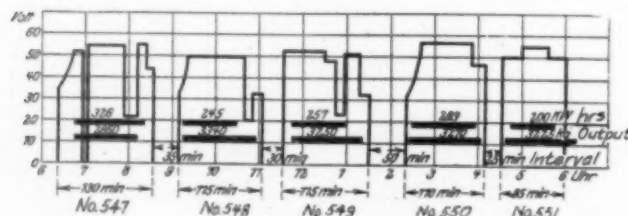


FIG. 13.—FURNACE OPERATION IN 12 HOURS.

(Desulfurization from 0.040 to 0.012%.)

with the radiation losses attendant upon charging, slagging and tapping.

Further, the above figures show that the cooling of the steel electrode requires only one-fourth of the amount of water needed for the cooling of the carbon electrode.

If the water flows at the rate of 5 liters per minute (1.3 gallons per minute), an equilibrium condition is established in the heat exchange as soon as the bottom steel electrodes have melted down to a point which is 20 mm (0.8 in.) below the level of the hearth. These liquid pole points which are gradu-

ally solidifying towards the lower end have the composition of the steel bath to be refined. No chemical change is thereby caused in newly charged metal, as they are emptied when the bath is tapped. The pits in the poles are then again filled with newly charged metal. The effect of cooling is a uniform fusing of the pole points, and a homogeneous current density through the same.

On account of the uniform current density and the uniform heating effect of the electric arc, the bath is heated on the hearth surface and the upper layers in such a way that a sticking of steel in the originally cold corners of the hearth is soon eliminated.

Electrical Conditions.

The quiet melting of a cold scrap steel charge has been recognized as a great advantage of the Girod furnace. Figs. 11 and 12 give the fluctuations of the kw consumption as plotted by a recording wattmeter, starting with a liquid charge. The regulation of voltage and current is almost always automatic; hand regulation (with the automatic regulator disconnected) is used only during slagging.

Influence of Temperature and Size of Furnace on Energy Consumption.

The furnace size, the length of intervals between successive heats, the cooling of the furnace during these intervals, the condition of the charge, the desired quality of the end products and other matters can change the energy consumption very considerably.

Fig. 14 shows the influence of the weight of charge in kilograms upon the specific energy consumption in kilowatthours per ton. The results are averages taken from 500 successive

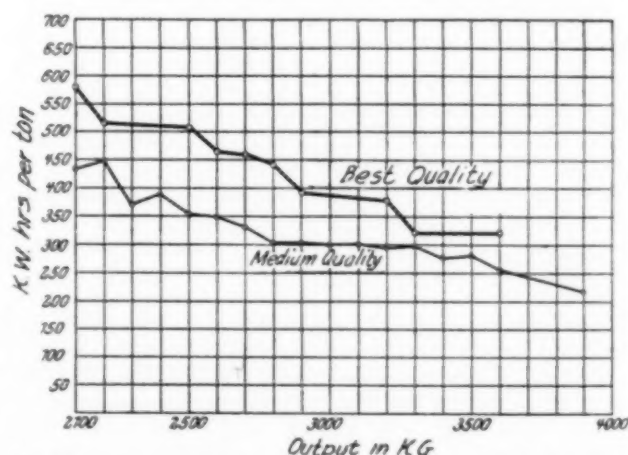


FIG. 14.—SPECIFIC ENERGY CONSUMPTION AS FUNCTION OF WEIGHT OF CHARGE.

heats, including alloy steels. The energy consumption depends a great deal on the weight of the charge and decreases in large furnaces.

The loss of heat through radiation and conduction as also the fluctuations of temperature decrease with increasing size of furnaces. The temperature is much more easily regulated in larger furnaces than in smaller ones.

The minimum economical weight of charge is about 2800 kilograms in Fig. 14, the specific energy consumption not changing materially with increasing weight up to 3300 kilograms.

The specific energy consumption depends still more on the temperature of the furnace and this depends greatly on the length of the intervals between charges. Fig. 15 shows the decrease of the temperature after tapping; this decrease is very rapid compared with that of an open-hearth furnace, the temperature of which sinks down to 1000°C after a 25-hours pause and which cools afterwards at the rate of 20°C per hour. Under such conditions it is possible to get a high thermal effi-

ciency with the Girod furnace only, if it is charged, at the latest, 20 minutes after the last tapping. The interior of the furnace is then still at about 1300° C. The energy consumption per ton is then 188 kw hours for medium-quality steels and 270 for high-quality steels.

Under very favorable conditions it was possible to refine 3654 kg (8039 lbs.) of steel with an energy consumption of 158 kw hours per ton, including refining by oxidation, deoxidation, and desulphurization. Large Girod furnaces are more favorable in this respect, as they can be charged immediately after tapping, i. e. while they are at a temperature 200 degrees higher.

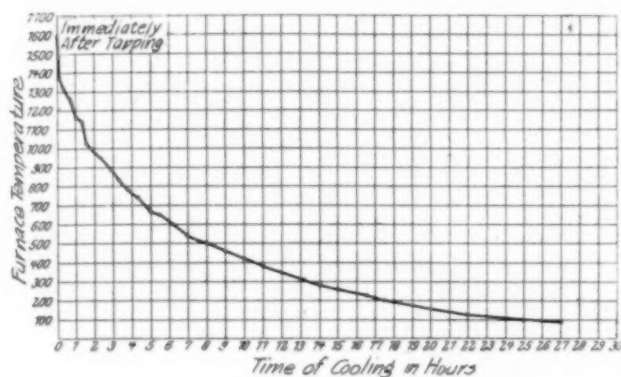


FIG. 15.—COOLING OF GIROD FURNACE.

Metallurgical Reactions.

Fig. 16 is a diagram of the chemical reactions during a heat in the Girod furnace and illustrates the changes of the carbon, manganese, phosphorus, sulphur, and silicon contents.

The results of the analyses of 19 samples are here plotted. Sample 1 is that of the charge, sample 2 was taken after addition of 15 kilograms of ore, sample 3 after addition of 50 kilograms of ore. Sample 4 and 5 were still taken during the period of refining by oxidation, sample 5 just before removing the oxidation slag.

Samples 6 to 19 were taken during the deoxidation period. After addition of 16 kilograms of Girod carbons and 5 kilograms of ferromanganese sample 6 was taken. After addition of 12 kg ferrosilicon (50 per cent Si) and 40 kg refining slag

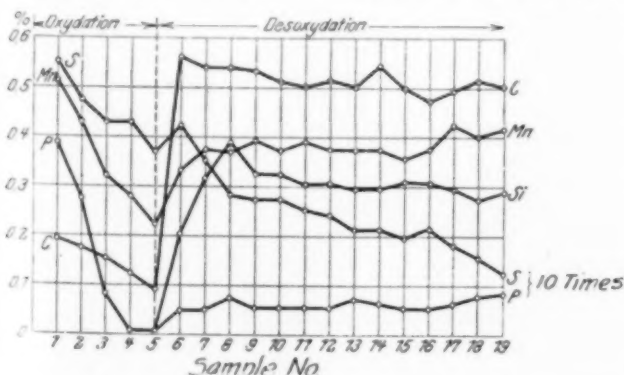


FIG. 16.—ANALYSIS OF C, MN, P, S IN 19 SAMPLES OF METAL DURING ONE HEAT.

sample 7 was taken. After addition of 4 kg ferrosilicon sample 8 was taken; after addition of 20 kg refining slag sample 9; after addition of 3 kg ferrosilicon sample 10; after addition of 2 kg powdered petroleum coke sample 11; after addition of 1 kg ferromanganese (80 per cent Mn) sample 12; after addition of 8 kg lime sample 13; nothing was added when sample 14 and 15 were taken; after addition of 10 kg lime and fluorspar and 2 kg powder petroleum coke sample 16 was taken; after addition of 2 kg ferromanganese sample 17; nothing was added after that, sample 19 being the final sample.

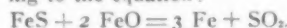
Oxidation Period.

The oxidation period must be preceded in many cases by a melting period of a length corresponding to the interval between charges, as a longer interval will make the steel solidify in parts. During this time ore is gradually added to the bath in the furnace, and carbon, manganese, phosphorus, and sulphur are oxidized.

The carbon oxidizes only slowly, in spite of the high temperature in the furnace. In the heat illustrated in Fig. 16, 0.10 per cent carbon is oxidized in 55 minutes, which is equivalent to 53 per cent of the original carbon content. Although the manganese is present in a more concentrated state than the carbon, only 58 per cent of the original manganese content is oxidized in the same time, which is equal to a decrease in manganese from 0.52 to 0.22 per cent.

Dephosphorization can be carried out to the furthest extent since the concentration of the oxidizing agent can be governed at will and the slag can be maintained very basic. Phosphorus is the easiest to remove of all the impurities in the charge. With progressing decarburization the phosphorus decreases steadily and the original low temperature of the bath favors the oxidation of the phosphorus. Already the third sample taken from the heat in Fig. 16 after a charge of 50 kg. ore, showed only 0.008 per cent phosphorus, against 0.039 at the start, which is a reduction by 80 per cent. Practically all phosphorus is removed before the slagging proceeds.

Sulphur decreases during this period of refining by oxidation, in Fig. 16 from 0.055 per cent to 0.037 per cent, which is a reduction by 33 per cent. Wüst claims that slag containing ferrous oxide dissolves iron sulphide to a certain extent. In stronger concentration it reacts with the ferrous oxide according to the equation:



The sulphur dioxide escapes and can be clearly recognized. But a part of the sulphur may still remain dissolved in the slag together with free ferrous oxide, as is shown by the sulphur of the oxidation slags of table III (p. 587). The figures in table IV (p. 587) show that the average decrease in sulphur during the oxidation period amounts to 26 per cent.

The oxidation slag with its phosphorus is drawn off and the last traces removed by mixing lime with it. This avoids rephosphorization of the bath by the carbon and silicon additions in the following period. One slag is sufficient to remove all phosphorus down to traces.

Deoxidation Period.

Now the deoxidizing slag is produced by adding to the bath about 20 kg. (44 lb.) lime, 3 kg. (6.6 lb.) sand and a like quantity of fluorspar, and also 1 kg. (2.2 lb.) ferrosilicon (50 per cent silicon) per ton of steel. This slag dissolves the ferrous oxide which has meanwhile been formed on the unprotected surface of the metal, and thereby becomes black.

The destruction of the ferrous oxide in the slag is indispensable for the deoxidation and desulphurization of the bath. For the deoxidation the silicon (from the added ferrosilicon) is not sufficient and 1 to 2 kg (2-4 lb) powdered petroleum coke are added to assure complete deoxidation. In this way a slag is obtained which in the air quickly disintegrates into a white powder; this indicates that the ferrous oxide has been reduced down to traces.

With hard charges an amount of petroleum-coke or waste carbon electrode pieces, in a quantity as required for the desired carbon content, are added to the bath for purposes of carburization, prior to the formation of the deoxidation slag. As soon as the slag is readily fluid and free from oxide, the additions are made of ferromanganese, ferrosilicon, etc.

The addition of these alloys which are often impure may increase the phosphorus and sulphur content considerably. Thus in Fig. 16 sample 6 shows that the phosphorus has increased from traces to 0.08 per cent, due to the addition of ferromanganese with 0.4 per cent phosphorus.

In following the curve for sulphur in Fig. 16 it will be seen that the amount increases during the carburizing process, showing that the bath has absorbed sulphur from the carburizing agent; this sulphur is then gradually removed from the bath by

TABLE I.—HEAT E 744.

		Sample					
		No.	C%	Mn%	P%	S%	Si%
Oxidation period. Charge.....		1	0.09	0.54	0.020	0.042	trace
After adding 10 kg ore.....		2	0.09	0.44	0.016	0.037	"
After adding 15 kg ore.....		3	0.09	0.34	0.010	0.037	"
After adding 20 kg ore.....		4	0.09	0.28	0.005	0.033	"
Sample just before slagging.....		4	0.09	0.28	0.005	0.033	"
Deoxidation period. After adding:							
22 kg coke, 60 kg refining slag.....		5	0.49	0.20	0.005	0.037	"
40 kg refining slag.....		6	0.53	0.24	trace	0.037	"
2 kg ferrosilicon, 20 kg refining slag.....		7	0.48	0.24	"	0.035	"
12 kg ferrosilicon; 18 kg ferroman- ganese.....		8	0.54	0.75	0.013	0.022	0.20
1 kg powdered lime.....		9	0.52	0.70	0.010	0.021	0.22
1 kg powdered petroleum coke.....		10	0.52	0.70	0.010	0.023	0.21
3 kg ferrosilicon.....		11	0.57	0.74	0.012	0.018	0.26
Final sample.....		12	0.58	0.70	0.015	0.014	0.24

Final product, 2985 kg electric steel.

Charge, 3030 kg soft steel.

During the refining period by oxidation sulphur was removed as follows:

3030 kg charge (0.042% S) contain..... 1273 grams S

3030 kg (0.033% S) just before slagging contain. 1000 "

Sulphur removed..... 273 grams S or 21.4%

After slagging 3000 kg steel (0.033% S) contain 990 grams S

Absorption of sulphur from 22 kg petroleum coke with 0.60% S..... 132 "

Sulphur in metal bath..... 1122 grams S

At the end of the desulphurization period (0.014% S)..... 420 "

Removed from metal bath..... 702 grams S

For the production of a deoxidation slag there were added:

80 kg lime with 0.08% sulphur..... 64 grams S

30 kg fluorspar with 0.07% sulphur..... 21 "

10 kg sand with traces of sulphur..... traces "

3 kg ferrosilicon with traces of sulphur..... traces "

The deoxidation slag contains..... 85 grams S

135 kg final slag with 0.5% S..... 675 "

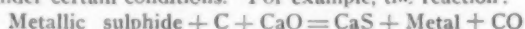
The slag has therefore absorbed from the bath 590 grams S

There were removed from the bath..... 702 "

Missing 112 grams S or 16%

the refining slag. There is a gradual decrease in the sulphur percentage up to sample 13, but no further decrease from sample 13 to 16. Desulphurization only starts again, after the thick highly basic lime slag has been rendered again effective by the addition of fluorspar (before sample 16).

The chemical reactions between the metal and slag occur only under certain conditions. For example, the reaction:



occurs only when the slag becomes white and readily fluid, i. e. after the removal of the metallic oxides down to traces. This is evident from table III (p. 587), which shows that in the slag the increase of sulphur runs parallel to the decrease of ferrous oxide and manganese oxide. It is wrong to assume that when a white slag free from metallic oxide is obtained (as is already the case with sample 12) the bath is completely desulphurized, and an interruption of the smelting process at this time would be a mistake, since the desulphurization has only begun and needs time to be complete.

As long as the bath is covered with an oxide slag it takes up oxide out of the slag and becomes thick and sluggish, which may be the reason for incomplete desulphurization in the presence of an oxide slag. When white slag has been formed on the metal bath, it absorbs the oxide from the bath and the oxide is destroyed by the carbon added and the calcium carbide formed under the electric arc.

The bath becomes more readily liquid through the removal of the ferrous oxide, and now the iron sulphide which is suspended in the bath will rise to the surface of the bath and be absorbed by the slag.

It has been observed that during the hottest period of the process, white vapors rise between the carbon electrode and the cooling ring which vapors deposit on the electrode in form of a mouldy, fibrous crust. As these deposits contain as much as 24 per cent SiO_2 , it may be assumed that these vapors consist chiefly of silicon sulphide SiS_2 . This gaseous compound is oxi-

dized by the air to SiO_2 and SO_2 and the white deposit of SiO_2 is obtained.

As the final finishing slag has only traces of manganese oxide, and no manganese sulphide in its composition, it is to

TABLE II.—HEAT E 784.

TABLE II.—HEAT 2789.							
	Sample	No.	C%	Mn%	P%	S%	Si%
Oxidation period. Charge.....	1	0.15	0.54	0.034	0.054	trace	"
After adding 10 kg ore.....	2	0.14	0.40	0.021	0.048	"	"
After adding 15 kg ore.....	3	0.14	0.34	0.016	0.044	"	"
After adding 25 kg ore.....	4	0.10	0.29	0.008	0.046	"	"
Just before slagging.....							
Deoxidation period. After adding:							
24 kg petroleum coke; 50 kg refining slag	5	0.50	0.26	0.010	0.034	trace	"
40 kg refining slag.....	6	0.50	0.26	0.010	0.038	"	"
30 kg refining slag.....	7	0.49	0.27	0.010	0.026	"	"
9 kg ferromanganese.....	8	0.50	0.49	0.011	0.026	"	"
1 kg powdered petroleum-coke; 5 kg ferrosilicon	9	0.52	0.52	0.015	0.026	0.07	"
5 kg ferrosilicon.....	10	0.52	0.52	0.012	0.018	0.14	"
4 kg ferromanganese.....	11	0.56	0.61	0.015	0.010	0.14	"
Final sample.....	12	0.56	0.62	0.015	0.010	0.14	"

Production, 3150 kg electric steel.

Charge, 3200 kg soft steel.

Sulphur lost during refining by oxidation:

3200 kg charge with 0.054% sulphur contain.... 1728 grams S

3200 kg steel with 0.046% S just before slagging contain 1472 "

Sulphur removed by oxidation..... 256 grams S or 15%

After slagging, 3200 — 30 = 3170 kg metal with 0.046% S contain..... 1458 grams S

Sulphur absorbed from 24 kg petroleum-coke with 0.75% S..... 180 "

Present in metal bath..... 1638 grams S

At end of desulphurizing period (0.010% S) there are present..... 317 "

Removed out of metal bath..... 1321 grams S

To form the deoxidizing slag there were used:

80 kg lime with 0.164% S..... 131 grams S

30 kg fluorspar with 0.07% S..... 21 "

10 kg sand with traces of S..... traces "

3 kg ferrosilicon with traces of S..... traces "

Deoxidizing slag contains..... 152 grams S

170 kg finishing slag with 0.89% sulphur contain 1513 "

Slag has absorbed from the bath..... 1361 grams S

Amount removed from metal bath..... 1321 "

Surplus 40 grams

TABLE III.—COMPOSITION OF SLAGS.

Chemical constituents.	Oxidiz. slags.	Deoxidizing slag		Finishing slag.
		After carburization.	After main charge.	
CaO	41.69	59.50	69.45	75.85
SiO ₂	9.58	26.78	17.90	13.20
FeO	28.36	1.02	0.27	0.13
MnO	0.99	1.44	0.52	trace
S	0.62	0.39	0.90	1.22
Fe ₂ O ₃	6.43	0.23
Al ₂ O ₃	1.73	1.67	1.27	1.73
MgO	6.99	5.54	5.78	4.22
P ₂ O ₅	1.47	0.07	0.12	0.09
Oxide of bases	20	20	22	22
Oxide of acids	9	15	10	8
Notes—	Black and streaky.	Grayish brown sand	White sand.	White powder.

TABLE IV.—DESULPHURIZATION EFFECTED DURING OXIDATION PERIOD.

	Ore added.	C%	Mn%	P%	S%	Si%	Decrease of S.
Charge E 152.....	15 kg	0.19	0.76	0.02	0.045	trace	26.7%
Sample 1.....	just before slagging	0.13	0.48	0.01	0.044	"	
Charge E 169.....	30	0.20	0.64	0.03	0.056	traces	30.4%
Sample 1.....	before slagging..	0.12	0.28	trace	0.039	"	
Charge E 455.....	5	0.23	0.62	0.030	0.059	traces	23.7%
Sample 1.....	before slagging..	0.22	0.44	0.010	0.048	"	
Sample 2.....	10	0.21	0.42	0.010	0.045	"	15.4%
Sample before slagging..	15	0.21	0.38	0.005	0.045	"	
Charge E 758.....	25	0.17	0.56	0.025	0.052	traces	15.4%
Sample before slagging..	25	0.09	0.22	traces	0.044	"	

TABLE V.—ANALYSES, TESTS AND APPLICATIONS OF VARIOUS ELECTRIC STEELS.

V#.	W.	Cr.	Ni.	C.	Mn.	P.	S.	Si.	TENSILE STRENGTH.			Applications.
									Kg.- Sq. Mm.	Elongation on 200 Mm. Bar.	Reduction of Cross Section.	
.....	0.13	0.54	Trace	0.014	0.16	41.2	30	65.6	Angle 130x65x8 mm.
.....	0.13	0.70	0.018	0.020	0.08	43	31	62.3	Angle 130x65x8 mm.
.....	0.19	0.56	Trace	0.010	0.13	47	31	55	Angle 130x130x12 mm.
.....	0.22	0.90	0.010	0.017	0.25	58	27.5	52.8	Iron 150x12 mm.
.....	0.25	0.88	Trace	0.017	0.26	60	23.5	49.1	I-beam N. P. 30.
.....	0.33	0.74	0.010	0.010	0.20	65	19	42	Engine and machine parts.
.....	0.61	0.62	0.010	Trace	0.47	86	13.2	23.5	50 mm. square blocks.
.....	0.53	0.68	0.008	0.010	0.37	86.9	12.8	30.3	Flat steel bars.
.....	0.57	0.70	Trace	0.008	0.40	89.4	10.7	21.4	Flat steel bars.
.....	0.72	0.42	0.020	0.010	0.34	97.1	6	23.8	Dies.
.....	0.82	0.48	0.006	0.010	0.25	99.5	5.8	20.1	Punches.
.....	0.70	0.72	0.010	0.010	0.36	83.8	11.5	19.8	Tramway car-wheel tires.
.....	0.47	0.78	0.010	0.012	0.24	71.5	17.5	32.9	Railway car-wheel tires.
.....	0.36	0.78	0.013	0.010	0.17	64	19	40.7	Axles.
.....	0.48	0.78	0.007	0.019	0.28	74.4	13.3	24.9	Locomotive wheel tires.
.....	0.52	0.78	0.015	0.018	0.19	78.7	11.1	20.5	Locomotive wheel tires.
.....	0.48	0.84	0.007	0.018	0.23	72	17.7	44	Shafting.
.....	72	16.5	45.2	Shafting.	
.....	1.89	0.26	0.78	Trace	0.012	0.12	61.8	23.5	48.6	Spindles.
.....	1.98	0.20	0.72	0.012	0.018	0.13	54	22	58.6	Spindles.
.....	1.04	1.02	0.58	Trace	0.012	0.16	Knives, shears, chisels.
.....	1.50	0.92	0.49	0.82	0.012	Trace	0.32	79.7	15	40.4	Tools.
0.19	0.85	0.46	0.60	Trace	0.012	0.22	79.6	10.2	32.9	Pieces for forging.
.....	62.1	33	59.5	Quenched at 700° C. in oil	
.....	59.6	36	63.8	Quenched at 700° C. in water	
.....	0.68	0.32	0.010	0.010	0.23	93.3	9	18.8	Rolls for making grooved rail

This includes the quantities of sulphur absorbed from the carburizing agents, etc., and again removed.

To sum up, the removal of the sulphur takes place in three different ways:

(1) During the oxidation period sulphur is removed through the action of the slag rich in ferrous oxide, whereby sulphur dioxide is formed which escapes.

(2) In the deoxidation period the white slag, free from ferrous oxide, removes sulphur.

(3) The third agent in desulphurization is silicon, forming silicon sulphide.

It is now generally believed that oxides, gases, and slag enclosures have a much more unfavorable influence on the physical properties of steel than phosphorus and sulphur below 0.02 per cent. The electric steel furnace allows the steel bath to rest a long time under the neutral non-oxide slag, which procedure can be carried out in the crucible only with difficulty without chemical changes, and which cannot be carried out at all in the open hearth. The possibility of removing the oxides, gases, and slag enclosure in this way is, therefore, an important advantage of the electric furnace.

As regards deoxidation, it might be said that it is started by introducing carbonaceous materials. The extent and the speed of deoxidation depend on the temperature, the amount and the action in time of the carbon additions. Complete deoxidation of steel with the aid of carbon alone (even with a slag completely free from ferrous oxide) is possible in the electric furnace only, if the refining period is extended for a very considerable time. It is, therefore, advisable to remove the last traces of oxygen by the use of other deoxidation agents like manganese and silicon in order to bring the oxygen into the slag. The oxygen compounds in the slag are reduced by the aid of ferrosilicon or powdered charcoal.

The calcium carbide formed at the greatest heat center under

the arc can be increased so much in amount by raising the tension of the electric arc that during the cooling of the slag ladle the acetylene is ignited with explosive violence.

Coussergues (*Revue de Metallurgie*, 1909, p. 589) claims that the calcium carbide causes the formation of acetylene at the contact surface of metal and slag according to the reaction: $\text{CaC}_2 + 2\text{H} = \text{Ca} + \text{C}_2\text{H}_2$, thus removing hydrogen from the steel and thus rendering electric steel more free from hydrogen than even crucible steel.

The products of the Girod furnace in the Gutehoffnungshütte are carbon steels of various degrees of hardness, also alloy steels, like nickel-steel, chromium steel, nickel-vanadium steel; chromium-vanadium steel, chromium-tungsten steel for high-class machine and engine parts and tools.

Table 5 gives a short synopsis of the results of tests and analyses, and the application of electric steels made in the Girod furnace.

Oberhausen, Rhineland,
Germany.

Cobalt.—The principal uses of cobalt in the United States are in making glass and pottery. A beautiful blue is given to glass by the oxide of cobalt. Sympathetic inks, according to a report of the United States Geological Survey, are made from cobalt acetate, chloride, and nitrate that are colored when heated or colorless when cold. This interesting phenomenon is due to the change in color of the salts on the absorption of water. When dry they are blue and easily seen on paper; when damp they are pink; and when dilute, colorless. A puzzling application of this principle may be in a doll whose dress is blue in dry weather but changes to pink when subjected to dampness, as in wet weather or when the doll is held in the steam of a tea kettle. Artificial flowers are made to show the same effect.

Progress of British Columbia Zinc Investigation.

The status of the investigation into the zinc resources of British Columbia, now being conducted by the Department of Mines, has been outlined in a recent report to the Director of the department by Mr. W. R. INGALLS, consulting engineer to the commission. The gist of the report follows:

Work has been prosecuted in the field of electric smelting at the metallurgical laboratory of McGill University, Montreal, under the immediate direction of Dr. Alfred Stansfield. A large number of experiments have been made with several forms of furnaces, certain of which have rather elaborate construction, and with a variety of material. The early experiments were directed chiefly toward a discovery of the metallurgical conditions which have hitherto prevented a satisfactory condensation of zinc as molten spelter. While I cannot say that these experiments have afforded us a complete explanation of these conditions, they have taught us a good deal; but in spite of the knowledge acquired we have been so far unable to master the difficulties.

We have indeed produced small quantities of spelter, and in some experiments have condensed a fairly large proportion as molten metal, but we have not yet been able to do that at will. Our experiments have thrown light on the principles of furnace design and have led us to condemn several types that we have tried. Our work has indicated that in order to achieve any material improvement over the ordinary practice of zinc smelting it is necessary to abandon certain features of the latter and contemplate continuous charging of the ore and reduction material, and discharging of the residuum without interfering with the process of distillation. These conditions introduce a multitude of perplexing difficulties which can be worked out only by tedious experimentation.

At the request of the secretary of the Canadian Mining Institute I presented at the meeting of the Institute at Quebec in March, 1911, a paper on the problem of mixed sulphide ores, that concisely summarizes the state of the art in the treatment of such ores, and the natural obstacles that block procedure in certain directions. This paper was published in the *Institute Quarterly Bulletin* No. 15, pp. 23-30, June, 1911.

A careful scrutiny of the work on the treatment of such ores that is done by other metallurgists and investigators has been maintained, and I have examined numerous proposals that have been presented with more or less detail, but I have not discovered anything save one that in my opinion holds out any promise of successful adaptation to the conditions existing in British Columbia. I regard the electric work as being of particular interest as an exploration in a virgin field of unknown possibilities. Doubtless with the same idea a great deal of work is being done in this field by numerous investigators in Europe and America. I have been informed within a few weeks that there are now two electrothermic zinc smelters in operation in Scandinavia, viz., one at Trollhättan, Sweden, using about 7000 hp, and one at Sarpsborg, Norway, using about 4000 hp. Operations at these works were inaugurated five or six years ago, but according to my information the results were for several years commercially unsatisfactory, and it is only recently that it has been claimed to have become possible to make spelter from ore upon an industrial scale. The companies operating these works maintain absolute secrecy, and I have not been able to learn any details of their operation.

Apart from the work in Scandinavia, as to the commercial success of which there is no information available, the electric smelting of zinc ore is, in spite of all claims to the contrary, not only still in the experimental stage, but is in the infancy of the experimental stage. Even if the metallurgical difficulties can be overcome, which is possible, I am of the opinion that no one, except perhaps the Scandinavians, is yet in a position to make any reliable estimate of commercial advantage, or commercial results in any way. It is, however, worth while to determine the possibilities and publish the results for the general benefit,

unveiling so far as possible the secrecy that is likely to be maintained as to the investigations in this field by private interests, having always in mind, of course, the hope that our work may develop a process that will be commercially applicable to the treatment of the zinc ores of Canada.

Cyanide Tailings Disposal in Mexico.

The discharge of cyanide tailings into streams, the waters of which are later used for domestic, industrial or other public purposes, brings up the possibility of legal complications between metallurgical companies and industrial or agricultural communities.

With a view to gathering information on the metallurgical side of the subject, the Instituto Mexicano de Minas y Metalurgia recently propounded a number of questions to leading metallurgical companies. The questions concerned:

(A) the percentage of cyanide contained in material discharged into streams, and the distance down stream at which these percentages persisted;

(B) the chemical or mechanical possibility of eliminating or reducing the quantity of cyanide in discharged tailings;

(C) the cost of conserving or impounding tailings in stacks or dams;

(D) the possible use of tailings for mine filling;

(E) the advisability and cost of purchasing adjoining lands which might be affected by the discharged tailings and cyanide.

From the replies received by the Institute and published in its *Informes y Memorias*, the following are given as representative opinions.

Mr. HUGH ROSE, of the Compania Beneficiadora de Pachuca, writes as follows: "(A) and (B)—I have no information of value on these subjects. (C) Where the location of a mill is such that a convenient dumping ground is available at reasonable expense, I consider it by all means advisable to retain the tailings in dumps or dams, according to the condition in which the tailings are discharged from the plant. Where the tailings can be discharged dry, they may be stacked either by belt conveyors or by cars. Where the tailings are discharged with considerable moisture, the South African practice of retaining the tailings by dams built automatically by the discharged tailings is an easy and cheap method. This method may be also used to recover the water in the pond so formed.

"The cost of conserving the tailings by either of the above methods naturally varies to some extent with the special conditions of each plant, but in any case should not be over 2 or 3 centavos per ton, exclusive of the cost of the land." (1 peso or Mexican silver dollar = 100 centavos).

"In a great many instances the conservation of the tailings is a wise policy, not only as a means of avoiding trouble with land holders, but in preserving perhaps for future generations a valuable asset. This applies particularly to the silver mills of Pachuca. There can hardly be any doubt about further advances in milling metallurgy making later economically available the values now discharged in the tailings.

"(D) The difficulties attached to using the tailings for filling stopes underground are increased where the tailings are very fine, as is the case with all sliming plants. It is not likely that this method can be adopted in general, as it requires such special conditions covering the location of mine and mill, method of mining, etc. It is specially applicable where the topography is flat or slightly rolling, as is more the case of the Rand, and in some of the flat mineral deposits.

"(E) The purchase of polluted lands would appear to be a very unwise policy, applicable to advantage only in special cases. Not only would it be difficult to secure the land at fair prices, but the adoption of such a policy generally would naturally result in complications to mills so located where the purchase of such lands would not be desirable or necessary.

"In conclusion, I would say that it is the intention of this company to impound its tailings by dams, as discussed in para-

graph C, for the purpose of not only avoiding controversies with land owners, but also of preserving the tailings for later treatment by this company or its successors."

* * *

Mr. D. L. H. FORBES, metallurgical engineer of the Tigre Mining Company, gives the following opinion:

"The cyanide plant of the Tigre Mining Company, which is now about to start operation, will discharge into the Tigre Cañon 250 short tons per day of slime residues containing 40 per cent moisture. These residues will carry the cyanide equivalent of about 50 lb. of potassium cyanide which will be wasted by the displacement of 42 tons of weak barren solution per day.

"The presence of this amount of cyanide with the residues will give the water in the Tigre Cañon a maximum solution strength of 0.025 per cent KCN during the dry season.

"While it is believed that such a weak cyanide solution would have its poisonous properties completely destroyed before flowing a greater distance than two miles, yet the Tigre Mining Company, in order to avoid the possibility of troublesome damage suits, has acquired a strip of land which takes in both sides of the cañon from the mill to the Bavispe River, a distance of ten miles. The writer is informed by Mr. L. R. Budrow, manager of the Tigre Mining Company, that the total cost of protecting the company's interests in this way was approximately 60,000 pesos.

"It is not the present intention of this company to collect the residues from the cyanide treatment in a dry condition, but the plant has been so designed that this may be easily accomplished whenever it shall be thought advisable, I would estimate that the extra operating cost for stacking slime residues which had been previously air-dried on the filter leaves would be 6 to 8 centavos per dry short ton. In the case of this plant it could be accomplished best by means of a 12-in. carrying belt. The initial cost of such a conveyor installation would be about 4000 pesos."

On the subject of filling the mine with tailings the writer is of the opinion that, aside from the inconvenience and expense of handling the tailings in this manner, such a method of disposal would be inadvisable at this property; both because of the method of mining used here, and on account of the timbering expense and danger that would be involved. As the veins of this property all have a nearly vertical pitch and are comparatively narrow, it has been found most economical to mine the ore bodies by a system of overhand stoping.

"Each block of ore is mined in such a manner that the stopes are kept nearly filled with broken ore until the level above the stope is reached. The ore is withdrawn in quantity only sufficient to compensate for its extra volume when broken, and the stopes are never entirely emptied until the section in which they are located is worked out and ready to be abandoned.

"Even if the filling of the mine with tailings were feasible, the fact that practically all precious metal ores are extracted from the upper workings of the mine first makes it obvious that the storage of the large volumes of mud in a pasty condition would have to be above the main roads and passageways and would therefore constitute an element of danger and an extra expense that would seriously handicap any mining enterprise that might adopt it.

"The practice of rendering escaping cyanide mill solution innocuous by special treatment has not been tried at any of the plants with which the writer has been connected. If such a procedure were made obligatory by statute, undoubtedly several methods would at once suggest themselves, the choice of which would depend largely upon local conditions and the spirit of the law.

"The judicious use of any one of the stronger mineral acids, while it would destroy the most active poison present in the solution, would also be adding another impurity to water that would still be unfit for drinking purposes. If, however, the application of the law should require the complete purification of the water escaping from the mill, a special procedure consisting

of precipitation of all salts in solution and subsequent filtration would have to be developed. Such an operation as the latter would involve expense that many companies which are now operating on low-grade ore and tailing dumps with small margin of profit could not bear.

"In most instances, however, it will probably be found that sufficient purification takes place in a small running stream. Wood, leaves, fibrous roots of plants and the carbonic acid present in the atmosphere are all well-known agents that decompose cyanide, and as these are always present in the smaller streams of this country it is evident that waste mill solutions mingling with the water of such streams must soon be rendered harmless in so far as their cyanide properties are concerned."

It should be remembered that the practice, which is becoming prevalent in the gold and silver mills of Mexico, of crushing ore in cyanide solution enables the operator to obtain a closed solution circuit, whereby practically no cyanide is run to waste unless "fouling" occurs. Wherever the ore is of such a character that the extraction of values is satisfactory and the decomposition of cyanide while in contact with the raw ore not excessive, crushing in solution is to be recommended, and it has the advantage of lessening the possibility of legal complications from actual or supposed harmful effects of residues escaping from the mill.

* * *

The following compilation of information from the company's several mills was contributed by Mr. GEORGE BRYANT, resident director of the Guanajuato Development Company:

(A).—*Peregrina Mill*, capacity 500 metric tons per day of tailings about one half sand and one half slime are discharged into a large reservoir which is opened once in two years and then during the occurrence of the heaviest rains, so that both the water impregnated with cyanide and the tailings themselves are taken away by flood water and washed a distance of from 20 to 30 km from the mill almost immediately. The superintendent's answer to question (A) is, that "the waters on leaving the mill contain about 0.03 per cent cyanide, and are practically unchanged on reaching the reservoir."

San Prospero Mill (Mexican Milling & Transportation Company), capacity 150 metric tons per day, discharges slimes immediately into the stream, where they join the drainage waters from the city. The sands, comprising one-half of the tailings, are stacked until the rainy season, when they are also carried away by flood waters. The superintendent states as follows, answering question (A): "The water leaving this mill contains on the average 150 grams cyanide to the ton. This proportion is greatly reduced about 400 m below the point of the discharge."

Pinguico Mill, capacity about 300 tons per day tailings about one-half sand and one-half slime. The slimes are discharged directly into a stream below the mill in which there is sufficient water for about one-half of the year to push them along, while during the dry half of the year they collect until the floods of the rainy season push them further down. The sands collect until the rainy season, when portions of them are washed away by the floods. The superintendent states, in answer to question (A): "The solution discharged from this plant with the tailings carries 0.02 per cent KCN. All traces of solution disappear 50 m below the plant."

Jesus Maria Mill, (Guanajuato Amalgamated Gold Mines Company), capacity 450 tons per day of tailings about one-half sand and one-half slime. The sands are stacked and some of them are carried away during the rainy season. The streams flow as long as there is water in the same to push them, which is about nine months in the year. The superintendent answers question (A) as follows: "The water leaving this plant flows down a cañon having very precipitous sides, and for some 4 km (1 km—0.62 mile) the total land contiguous to the stream and used for agricultural purposes does not total one hectare. From that point on there are orchards and more or less cultivated land."

Upon leaving the company's property the solution titrate is, in terms of

	KCN	and	CaO
	0.03%		0.31%
1 km down stream	0.03%		0.26%
2 " " "	0.02%		0.19%
3 " " "	0.015%		0.014%

These samples were taken at five-minute intervals over a period of two hours, which represents a complete cycle on our filter, and gives the average solution discharged. It includes the cyanide of sodium, potassium and the simple cyanide of zinc (ZnCy₂), usually estimated as free cyanides.

The total cyanides include KCy.KSCy, K₂FeCy₄, K₂ZnCy₄, and the related sodium cyanide compounds are as follows:

Total cyanide in terms of KCN:

At plant	0.05%
2 km down stream	0.04%
4 km down stream	0.025%

(B).—Three of our superintendents answer that they do not know of any practical chemical or mechanical process for further reducing the small proportion of cyanide contained in the discharged water and tailings. A fourth superintendent answers as follows:

"It is doubtful whether sodium cyanide is especially detrimental to water used for agricultural purposes, as the sodium cyanide undergoes slow decomposition, giving off ammonia and urea (both of which are used in the manufacture of artificial fertilizers), and formic acid. Rather the excess of lime is the damaging material, and this could be neutralized by commercial acid, or possibly by our mine waters, should the pumps work continuously.

"Some years ago one plant let its waste solutions percolate through refuse matter from slaughter-houses, manure and sawdust to render them harmless, but with indifferent success. Then, for some years following the introduction of cyaniding milling in many camps, efforts were made to render the effluent solutions harmless, but I am not aware of marked success attending a single effort, and cannot recommend a process to effectively accomplish that result."

(C).—Peregrina answers that the case of discharging dry tailings would increase the milling cost by 1 peso per ton. Pingüico answers that 60 per cent of the tails discharged are perfectly dry and are washed away each year by the rains. Another answers, "About 45 per cent of our tailings are now discharged dry and do not enter the river. It would be impracticable to discharge the balance dry." I quote verbally from a fourth superintendent's answer: "The slime discharge averages 42.5 per cent moisture at present, our sand is dry for practical purposes. Should the tailings be discharged with a lower moisture per cent than our ore carries, the solution would accumulate by reason of this excess moisture until such a time as all storage capacity would be occupied, and then the excess day by day would run to waste. On the other hand, if not drier than the ore, there would be a continuous seepage of solution rather high in KCN from around the lower edge of the dump, and this would gradually find its way into the stream. Thus, while the objectionable elements would be lessened, it would not cause their elimination."

(D).—At all of our plants, mines and mills are so far separated that use of tailings for filling stopes would be certainly out of the question.

(E).—So far as we know, no damage is done by our tailings to agricultural lands, and up to the present time no claims have been suggested for such damage.

In only one case have I taken this matter into consideration. In that case we found that a total outlay of 46,000 pesos would be necessary to accomplish the purpose suggested in question (F), and figuring the ore supply in that property as 720,000 tons, sufficient for five years' work, we calculated that it would increase the milling cost by 6.4 cents, Mexican currency, per ton.

Possible Reduction of the Power Consumption in Electric Steel Refining Furnaces.

BY CARL HERING.

The question of the power consumption in electric furnaces for steel refining is, of course, of great importance and is sometimes even the criterion, hence it deserves the most careful consideration. The present article is an attempt to determine from present practice approximately what the maximum economy is which may be expected in a furnace designed with that end in view. Such a figure would show what the limit of economy is beyond which we cannot expect to go and how near present practice is to this best economy. As the deductions are based on data from an actual furnace they, of course, are subject to the same inaccuracies as the original data. But even if these data are not accurate, and the writer assumes no responsibility for them, they are not likely to be far wrong, and the description of the method of making the deductions may itself be of interest, as it can be applied in the future when better data become available. Even if the present data should have been 100 per cent greater (worse) the results deduced would still be surprisingly favorable.

The chief function of a furnace for steel refining is merely to keep the metal at a constant temperature during the process of refining. In the large Chicago furnace the metal is said to even leave at a slightly lower temperature than that at which it entered. Hence the only energy to be considered here is that necessary to keep the temperature constant; if, therefore, there are any chemical reactions which consume energy, or if there is any melting done, or any increase in temperature, the energy necessary for this must be added. Moreover, the heat lost through the doors, electrodes, etc., is also neglected, hence must be added in an actual case.

When the temperature is maintained constant it means that the total energy supplied to the charge is equal to that lost by it. The heat loss from the charge must, of course, all flow through the bounding surfaces of the charge—that is, through the top, sides and bottom; it is lost by radiation from the top and by conduction from the sides and bottom. For want of better data it is here assumed that the loss by radiation from the top is at the same rate per unit surface as that by conduction from the sides and bottom. Though this is no doubt not strictly correct, it is probably sufficiently so for the present purpose of reaching an approximate figure; this figure can be corrected later when better data become available.

It is also assumed in the present deductions that the insulation of the walls in actual practice is uniform for all different sizes of furnaces—that is, that the heat energy passing out of the metal per square foot of its outside surfaces is a constant and is equal to that in the particular furnace from which the data were obtained. If this insulation can be improved the final figures will be lowered accordingly.

Under these circumstances the heat losses through these bounding surfaces for a constant inside temperature are, of course, dependent only on the extent of this surface and not on the quantity of the metal within the walls.

If, therefore, the loss is a constant per square foot of surface of the metal, and is independent of the quantity of metal, it follows that the loss will be least when the metal is in the shape of a sphere. But as this is impracticable it will be assumed here that the ideal form in practice is a hemisphere, leaving a dome of an equal hemisphere above for working space.

The problem therefore is, on the basis of a constant loss per square foot of metal surface, what will be the total loss when the body of metal has a hemispherical shape?

This constant loss may be deduced from a recent paper by Mr. Albert Hiorth, read at the recent Toronto meeting of the American Electrochemical Society¹, in which he states that by actual test it required 180 kw to keep the steel melted over

¹See preceding issue of this journal, p. 534.

night in one of his five-ton, two-ring, single-phase induction furnaces for steel refining. As there are no electrodes, this represents the power which is being continuously lost through the walls. When the furnace was running it took somewhat more, but presumably cold slag was then being melted, and probably the covers were frequently removed; there was also presumably an increase in temperature; these doubtless account for the increased input of power. For the present purposes, therefore, it is probably fairest to use the figure for the all-night run. As these measured results were used by him to design a 30-ton furnace, it may be assumed that they are fairly correct.

He also gives all the dimensions of the crucible and body of metal; hence, one is enabled to deduce from them the outside surface of the charge of metal and the total outside surface of the crucible. These deductions have been made by the present writer and the result is that the average continuous loss of heat from the bonding surfaces of the metal (sides, bottom and top) is equal to about 13 kw per square meter, or

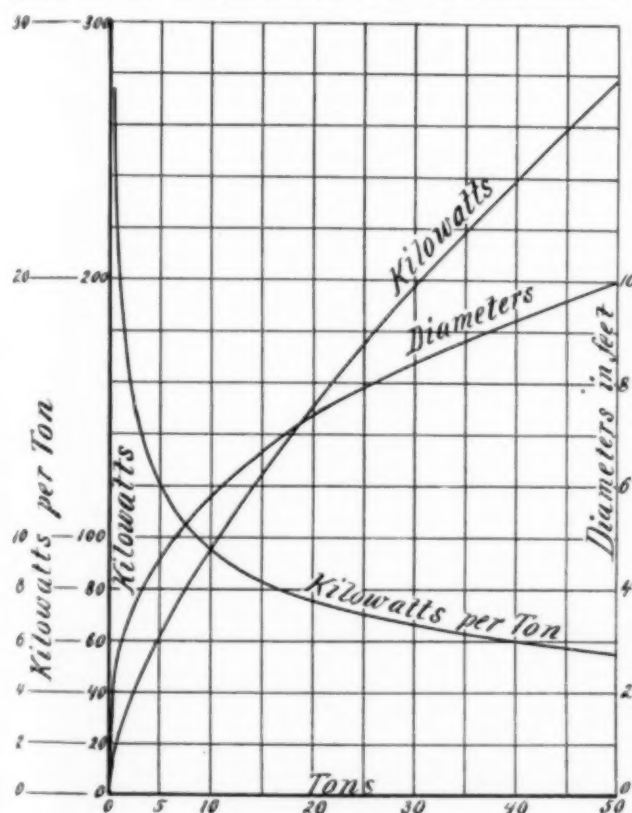


FIG. 1.—MINIMUM KILOWATTS PER TON AS FUNCTION OF SIZE OF FURNACE (CONDITION OF LEAST LOSS).

1.2 kw per square foot. The total surface of the charge was 13.9 sq. m., and the total loss 180 kw. The total exposed outside surface of the crucible was 56 sq. m., or about four times the metal surface.

This average figure, 1.2 kw per square foot of metal surface, is taken as the basis of the following deductions. It is doubtless greater in some parts and less in others, but, as the results are necessarily only approximate, a further refinement is hardly warranted.

In a Roechling-Rodenhauser single-phase, two-ring, induction furnace, the data for which was kindly furnished to the writer for this purpose, the surface of the metal charge was found to be 6.36 sq. meters, and the power necessary to maintain the temperature was given as about 80 kw.; it was a 2-ton furnace, hence considerably smaller than the other one. This reduces to a loss of 12.5 kw. per square meter of metal surface, or 1.17 kw. per square foot. The agreement is heretofore confirmatory and is in fact strikingly close.

As these figures can no doubt be improved, an easily remembered rough-and-ready figure for the losses through walls under steel refining conditions is therefore one kilowatt per square foot.

A ton (metric ton equal approximately to a gross ton) of molten iron occupies a volume of about 5.13 cu. ft. From this and the above data the following table has been calculated. It gives the kilowatts which would be required on the above bases to keep steel at the refining temperature, the bottom of the hearth being hemispherical and the metal just filling the hemispherical part. Hence it gives the least number of kilowatts, or kilowatts per ton, required under the best practicable conditions for obtaining the least loss. The curves give the same results graphically.

Capacity.				
Gross Tons.	Cubic Feet.	Diameters in Feet.	Kilowatts.	Kw. per Ton.
½	2.57	2.14	13.	26.
1	5.13	2.70	20.	20.
2	10.3	3.40	32.	16.
5	25.7	4.61	60.	12.
10	51.3	5.80	95.	9.5
15	77.0	6.65	124.	8.
30	154.	8.38	200.	6.6
50	257.	9.93	278.	5.5

The figures in the last two columns will be found to be far smaller than those of present practice, showing that there is a possibility of very great improvement in reducing the consumption of energy. The actual figure for the Hiorth 5-ton induction furnace was 36 kw per ton as against a possible 12 kw. In the South Chicago 15-ton furnace it is given as 50 against a possible 8.

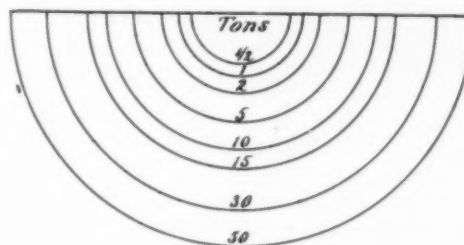


FIG. 2.—INCREASE OF SIZE OF FURNACE WITH INCREASE OF CAPACITY.

The figures in the table are based on the actual results in an induction furnace, but, as the heat insulation per square foot of metal surface of a simple hearth could no doubt be made still higher, the numbers in the last two columns can probably be still further reduced.

These extremely low power consumptions in the table point out the direction in which progress becomes possible if the power consumption is important. They can, of course, be expected to be reached only in those types of furnaces in which it becomes possible to give the hearth its best possible shape and in which the necessary slagging surface is obtained by energetic circulations, as, for instance, in the type of furnace devised by the writer and described in a recent paper read before the American Electrochemical Society² and abstracted in the May issue of this journal, p. 277. In other types the results may be approached more or less closely by so designing the hearth or crucible that the bounding surface of the metal is the least possible in that particular type.

The above deductions have nothing to do with the length of time of the refining. If by any improvements, such as rapid circulation, which brings all parts of the metal into repeated contact with the slag, the time of treatment could be reduced, it would, of course, reduce the energy required still more, and in direct proportion; this will reduce the kw-hours, as distinguished from the kilowatts, and it is with the latter only that the above deductions are concerned.

²Trans. Amer. Electrochem. Soc. Vol. XIX, 1911, p. 255.

At half a cent per kw-hour it would therefore cost only about 5 cents per hour and per ton of metal to keep it melted for refining purposes in a 10-ton furnace, or 3.3 cents in a 30-ton furnace. As stated above, any power used for melting cold slags, raising the temperature, supplying the losses through open doors, electrodes, etc., must be added hereto. These surprisingly low figures are a good illustration of the improvements which may be expected by a more careful and more rational design, with a view to obtaining the best heat economy.

The following further deductions indicate that even these low figures are probably too high, and that it may be possible to reduce them considerably, possibly to about half these values. In an induction furnace the walls of the crucible which surround the iron cores, are necessarily restricted in thickness; in the one from which these data were taken these surfaces are furthermore water cooled in order to protect the core; moreover, there are 1.65 tons of copper in the primary turns, which are intentionally placed as near as practical above and below the hot metal secondary, to increase the power factor; such a mass of copper will carry off a large quantity of heat. It is likely therefore that the average loss of 1.2 kw per sq. ft. of metal surface in this particular furnace is higher than it would be in one in which the wall thickness is not restricted and in which there is no such mass of copper near the charge. The outside walls of this crucible were about 23 in. thick (60 cm.), the bottom apparently about the same, while the inside ones were only about 14 in. (35 cm.); about one-quarter of the total metal surface was exposed to these thinner walls. The metal charge had about three times the surface exposed to the losses that it would have in a hemispherical hearth.

Taking a five-ton furnace with a hemispherical charge as an illustration, and assuming the wall to be 24 in. thick, the surface on the outside corresponding to 1 sq. ft. on the inside is about 3.5 sq. ft.; hence the flare is about the same as the average (4) in the Hiorth furnace. The mean cross-section of the path of the flow of heat from 1 sq. ft. of the metal surface will then be 1.87 sq. ft., namely, the geometric mean. The thermal resistivity of fire brick deduced from Wologdine's results^a is 22.4 thermal ohms for one inch cube. Hence the thermal resistance of such a frustrum of fire brick is $24 \times 22.4 / 1.87 \times 144 = 2$ thermal ohms; hence if the difference of temperature between the inside and outside is about 1400°C., the flow of heat $W = T/R$, would be $1400/2 = 700$ watts, as compared with 1200 watts deduced from the Hiorth furnace, or only a little over half as much; hence the figures in the table are likely to be nearly double what they should be. Moreover, the Wologdine values for the thermal conductivities are believed to be too high (100 per cent for fire brick, according to Clement & Ely), and if this is true, the flow will be considerably less than 700, perhaps only half as great, in which case the figures in the table would be reduced in the ratio of 1200 to 350, or to less than a third. It is likely, however, that magnesite was used as the lining, and this would increase the flow somewhat, as it conducts far better than fire brick. The rough agreement of the results deduced by these two entirely independent methods shows that the figures are not likely to be far wrong, and are no doubt on the safe side.

An additional check of the order of magnitude of these figures may be obtained from steam heating practice. The amount of heat emitted per square foot of surface of an iron heater, due to radiation and moving air, reduces to about 200 watts when its temperature is about 150°C., and it is said to be about 30 per cent greater from a sandy surface, hence 260 watts. The rate at which the heat would be emitted from the outside surface of the five-ton hemispherical furnace mentioned above is $700/3.5 = 200$ watts per sq. ft., using Wologdine's value, and $1200/4 = 300$, using Hiorth's results, thus again showing very good agreement considering the crudeness of the figures.

^aThis journal, January, 1911, p. 13.

It, therefore, seems probable that the surprisingly low figures in the table are not only safe, but will in practice turn out to be still lower, and perhaps considerably so, showing the great advantages that may be gained by first finding out what is the best possible design and then approaching it as closely as conditions permit. In an arc furnace the conditions are such that these very high economies can never be expected, owing to the fact that much of the heat of the arc never reaches the metal itself; the above figures refer to the losses of heat from the charge itself, and therefore apply more particularly to resistance furnaces.

Gold, Silver, Copper, Lead and Zinc Production in Western States in 1910.

The United States Geological Survey has recently issued reports showing official estimates of the production of the principal metals in Western States. For convenience the totals have been tabulated as below.

Arizona showed an increased output of gold, but decreased production of silver, copper, lead and zinc. In Colorado the output of lead and zinc increased, but gold, silver and copper showed a decrease. Idaho had an increase in production of silver, lead and zinc, but showed a decrease in gold and copper. In Montana there was an increase in lead and zinc, but a decrease in gold, silver and copper. Nevada had an increase in output of gold, silver and copper, but a decrease in lead and zinc. In Utah there was an increased output of copper and zinc, but a decrease in gold, silver and lead.

	Arizona.	Colorado.	Idaho.
Gold, value.....	\$3,149,366	\$20,507,058	\$1,096,842
Silver, fine oz.....	2,566,528	8,509,598	7,369,742
Copper, lb.....	297,491,151	8,359,535	7,037,292
Lead, lb.....	2,696,151	76,071,273	228,258,839
Zinc, lb.....	5,484,899	77,089,648	5,603,114
Total value.....	\$42,731,519	\$33,673,879	\$16,316,196
Tons mined.....	3,914,969	2,434,664	1,786,174

	Montana.	Nevada.	Utah.
Gold, value.....	\$3,730,486	\$18,878,864	\$4,032,085
Silver, fine oz.....	12,162,857	12,479,871	10,466,971
Copper, lb.....	284,808,553	64,359,398	127,597,072
Lead, lb.....	4,106,292	4,871,130	123,597,072
Zinc, lb.....	31,638,184	2,707,071	16,367,104
Total value.....	\$48,358,253	\$34,152,148	\$32,199,185
Tons mined.....	5,079,446	3,655,775	6,389,398

Sixth Congress of the International Association for Testing Materials.

The sixth congress of the International Association for Testing Materials will be held under the patronage of the President of the United States at the Engineering Societies' Building, New York City, during the week beginning September 2, 1912.

The nature of the work for which the congress will meet is concisely described in section 2 of the by-laws of the association as "the development and unification of standard methods of testing; the examination of the technically important properties of materials of construction and other materials of practical value, and the perfecting of apparatus used for this purpose."

Another of the most important functions of the association is the establishment of standard specifications for materials used in manufacture and construction. As these specifications are of greatest value to engineers, it is to be hoped that many American engineers will join the association in order to attend the congress and aid in establishing these specifications.

Prof. Henry M. Howe is the president of the American

Society for Testing Materials and acting president of the International Association for Testing Materials. Prof. Edgar Marburg, of the University of Pennsylvania, Philadelphia, Pa., is the secretary of the American society.

Mr. H. F. J. Porter, 1 Madison Avenue, New York City, is the secretary of the organizing committee of the congress, and all communications concerning the congress should be addressed to him.

An Historical Note on Ore Treatment in San Juan County, Colorado

BY WARREN C. PROSSER.

It is often stated that milling practice in San Juan County is far behind the times, which is probably true. Pressure has, however, been brought to bear within the past few years, by adverse circumstances, which has created the incentive to do better work, improve conditions and institute the best methods of ore treatment. This will no doubt result in successful modern practice at no distant future time.

In picking out the flaws of past and present methods the main fact discovered is that instead of having one problem to solve in arriving at successful extraction of the metallic values, such as might confront a camp mining one general character of ore, there are here a dozen to solve for each character of ore mined.

The early discoveries made in 1861 were abandoned necessarily on account of the violation of treaties with the Indians and it was not until 1873, when the Brunot Treaty was made, that mining operations started in earnest.

The year previous the Little Giant Company was organized in Chicago, anticipating the signing of this treaty, and they immediately took steps to replace their arrastres by stamps and amalgamation plates. Their ore was a brown quartz carrying free gold in quantity.

In these early days silver was high in value, ranging around \$1.29 per oz. and ores were mainly sought which carried high silver values. Gray copper and galena ores were found abundantly as superficial ores.

Mineral Point, where outcrop some of the largest and most extensive veins to be found anywhere, received much attention during 1874 and the year following. The pay streaks were large and some of the ore carried silver in hundreds and thousands of ounces to the ton. Gray copper ores were found in many properties and some rich pockets were opened. The Bill Young mine, for instance, produced \$50,000 in one pocket. The Old Lout produced an ore carrying bismuth and associated with a pyrite of botrioidal crystallization which ran \$3,000 per ton. The Alaskan mine produced \$100,000 in one pocket of this same character of ore.

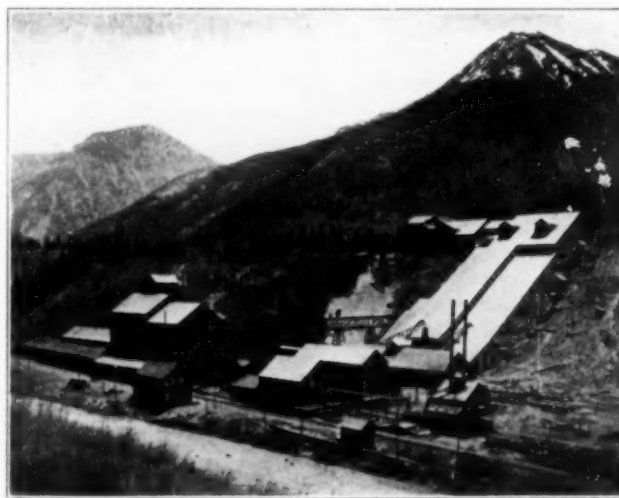
Early Silver Processes.

The most enterprising of these Mineral Point mines, however, was the San Juan Chief. It was early discovered that much of the value of the ore was being dissipated in excessive freight and treatment charges, and regretting that they did not know of some method of saving the gold, copper, lead and zinc, they decided to institute a method of treatment for silver alone. Before this some of the ore had been treated in a custom mill at Animas Forks with crusher, rolls, jigs and tables but the ore was slimed and great losses resulted. A buddle was put in but this too proved a failure.

Kempton and Thatcher finally built a small testing plant near Mineral Point, where they worked up a leaching process, supposed now to have been designed after the Augustin process, and then they built a larger mill which is now the San Juan Chief mill. Here were placed a Blake crusher, rolls, stamps, roasters and vats. The material from the mine, after being crushed and stamped, was dried in one of the roasters and then given a chloridizing roast in the other. It was then

fed hot into a vat of sulphuric and nitric acid solution where it was leached for thirty or forty hours. The solution was then drawn off and run through granulated copper suspended on a cloth screen immersed in water just below the top of another tank. The silver was precipitated by the copper which in turn was recovered by being precipitated on iron sheets hung in the tank below. The silver remaining on the screens was then filter-pressed, melted in large graphite crucibles and cast into bricks. Later it was attempted to crush and roll dry, and still later to concentrate on Cammett tables and canvas.

At Gladstone where the high grade gold-silver ore of Poughkeepsie Gulch was being delivered, and at Crookes mill, two miles north of Silverton, operating on the gray copper ore of the North Star mine on King Solomon Mountain, lixiviation works of the same character were installed a short time fol-



SILVER LAKE MILL AND TERMINAL EQUIPMENT TWO MILES ABOVE SILVERTON, COLO.

lowing the San Juan Chief operations. It is clearly evident that the method was designed to treat the gray copper ores, tetrahedrite, freibergite and perhaps stromeyerite, which were subject to excessive sliming by ordinary wet concentration.

The causes of the failure to treat the Mineral Point and Poughkeepsie gulch ores were varied. The freight on acid and salt was extremely high; the exact figures are not now available but one of the operators of the San Juan Chief tells that it was cheaper to buy a four horse team and wagon in Denver and haul it in. It was thought that the railroad considered the acid so dangerous that in case of a wreck the rails for miles around and the two nearest stations would be entirely destroyed. Again the failure to receive payment for gold and the absolute waste of the copper and lead values worked another hardship. Then, too, the presence of much zinc mineral interfered with successful concentration and reduction.

Smelting.

In 1874 the Green smelter, just north of Silverton, was erected and it was blown in the following year, treating principally the sulphide ores of Arastra Gulch and Hazelton Mountain. This is reported to be the first successful water-jacket furnace in Colorado.

Crooke and Company erected a smelter at Lake City in 1875, and drew sulphide ore from the mines in the northern part of the county around California, Mastodon and Poughkeepsie Gulches and Mineral Point. Mather and Geist of Pueblo also bought ore around the county in competition with the Crooke smelter.

In 1876 a concentrating mill was erected at Animas Forks to treat the ores of the Red Cloud mine near Mineral Point, but this was not successful.

Jas. Cherry built the Eclipse smelter at the mouth of Grouse Gulch in 1880, at a cost of \$65,000 and worked only a few tons of lead ore from the Mountain Queen mine.

The Green Smelter about this time was moved to Durango, where it became successively the New York and San Juan Smelting Company, The Durango Smelting Company, The San Juan Smelting and Mining Company, until 1895, when it became the Durango Plant of the American Smelting and Refining Company, now operating under that name.

In 1883 sampling works were operating in Silverton, under the management of E. T. Sweet, T. B. Comstock and Company, and Stoiber Bros. Two years previously the rich copper-silver deposits of the Red Mountain district had been discovered and a smelter was built by Thos. Walsh at Silverton on the site of the Martha Rose smelter ostensibly to handle these ores.

The samplers assisted in securing the production of great quantities of sulphide ores and a large tonnage of low grade ore of this character was developed. Their concentration was attempted by Edward G. Stoiber and Judge J. H. Terry.

Concentration.

Stoiber erected the first Silver Lake mill in Silver Lake Basin where forty stamps were followed by concentration tables. In 1893, a Bleichert tramway was built from this mill to the railroad and in a short time a 100-ton mill was built at the lower terminal, to which milling operations were then removed. In this latter mill were used rolls, jigs, vanners, tables, and later, canvas tables. The mill was burned in 1905, and in 1907 the present modern mill was built on the same ground.

J. H. Terry's operations were first carried on with a ten-stamp mill near Lake Em and the Sunnyside mine. Later a larger mill was built at the forks of Eureka Creek, two miles below and still later a forty-stamp mill was built at Eureka on the railroad, aerial tramways being run to the mine. This mill has been in continuous operation for many years.

The successes achieved by Judge Terry and J. H. Stoiber were emulated by many, some being fairly successful in concentrating and reducing their ores and others using the opportunity thus presented to secure the erection of mills on their properties with no ore in sight. The Gold King, an eighty-stamp mill; North star, twenty stamps; Hercules, twenty stamps; and the Iowa, a roll mill, were highly successful.

The year 1906 witnessed a mill building boom in San Juan County. The Old Hundred mill was built at enormous cost, using 1350 lb. stamps, plates, Card tables, and Sperry slimers. The slimers were soon discarded and amalgamation became their chief effort. Successful concentration was defeated through lack of classification, principally. At the Mogul mill the character of the ore, carrying high percentages of zinc, was the sticking point and the mill closed down after some very expensive work including the installation of Blake machines. The problem here remains to be solved to-day.

In 1905 and 1906 the Gold Prince mill was built at Animas Forks. This is a 500-ton mill of 100 stamps. It was built at a cost of half a million dollars, of steel and reinforced concrete and is a very modern building. It is hard to state just why its operation was not continuous but it is maintained that the ore reserves in the mine were not adequate to meet the enormous tonnage required. It was operated under receivership during 1909-10 and is now awaiting the outcome of litigation.

Very little data is available covering the milling of the sulphide ores. The savings were poor as a general rule. Several factors crept in to render the extraction difficult. A desire to stamp for purposes of amalgamation and to produce concentrates simultaneously, met with a loss of lead and other values in the tailings on account of the fine grinding. The presence of zinc, for many years a bugbear on account of the penalty inflicted by the smelters for its presence, induced further losses of included values upon being eliminated. A rhodonite gangue

introduced a waste mineral of high specific gravity, which crept up on the table and interfered with the separation. The Sunnyside mine has had all these factors to contend with. Here an attempt was made to produce a zinc product with Blake machines, which was partially successful, up to the time that the zinc plant was burned.

Prospects for Cyanidation.

The treatment of the siliceous ores of the county, carrying values principally in gold and silver, is to-day presenting a broad field for operation in which past efforts have been mainly experimental. In Ice Lake Basin an attempt was made to treat the dry ores of the overlying volcanics by amalgamation. The gold is coarse and rusty, however, and a saving of about 40 per cent was all that could be obtained. Cyanide tests on these ores gave an extraction of 92 per cent of the gold and 66 per cent of the silver, in eight hours, using compressed air for agitation and aeration. Only a small loss in cyanide was experienced.

On the Eastern Range, including a rather large area, the siliceous ores containing values in free gold associated with argentite were overlooked for many years save for an occasional shipment of high grade ore. The Intersection five-stamp mill is the pioneer in this field and this was built solely for experimental purposes. By amalgamation and the production of a pyrite concentrate on an Overstrom table a saving of 86 per cent of the combined values has been made. That it is a cyaniding field is unquestionably the case. The Esmeralda, Little Maud, Intersection, Ridgeway, Buffalo Boy and Crystal Lake properties have all had highly successful tests showing a good extraction with a small loss in cyanide.

If the reports concerning the extraction of values in some of the mills are true it is not surprising that few reliable records have been kept; but at the same time it must be borne in mind that a great complexity and diversity of ores exists in this district which has tried the skill of many good mill men. Moreover they had at their disposal very little machinery which could do the work, up to a very recent time.

The introduction of electrostatic machines has been the most important step that has been attempted for many years, and that it will be successful in the production of high grade products has already been proved by exhaustive tests, chiefly on zinc-bearing ores. The success of mining operations in the San Juan is now largely dependent on ore treatment, and the backwardness of milling operations has constituted unwittingly a conservation of a greatly diversified mineral wealth.

Silverton, Colo.

Reinforced concrete jigs have been decided upon to replace wooden jigs at the mill of the Cananea Consolidated Copper Company, Cananea, Mexico. They were designed by F. J. Strachan, superintendent of concentrators, and are described in detail in the *Engineering and Mining Journal*, Oct. 7. The inside of the structure is coated with a mixture of lye, alum, cement and water.

The so-called bicarbonating process of treating oxidized or carbonate ores is carried out by dissolving the minerals in a water solution of CO_2 under pressure, forming bicarbonates of the metals. The water solution is separated from the gangue, and the different mineral salts are caused to precipitate by successively reducing the pressure to the critical points at which the various carbonates are claimed to separate out of the solution. It is claimed, for example, that with copper and zinc in solution as bicarbonates under pressure, if the pressure is gradually reduced copper will precipitate before zinc, and that this means of separation can be made quantitative. The whole process is carried out in the cold, and consists of a system of agitating and pumping, with recovery of the CO_2 when the pressure is released from the bicarbonate solutions and when the precipitated carbonates are calcined. The idea has never been carried out commercially, but has been the subject of experimentation by H. D. Rankin and E. H. Westling.

The "Paragon" Electric Furnace and Recent Developments in Metallurgy.*

BY JOH. HÄRDÉN.

Recent developments in the art of steel refining in the electric furnace have called for certain modifications in the design calculated to meet the requirements of the metallurgist.

A retrospective view of the electrometallurgy of recent years will make this clear.

When the late Dr. Kjellin, whose untimely decease we all had to lament on last New Year's eve, designed his induction furnace, about 1894, to 1898, his primary object was to create an efficient melting machine for high-class crucible steel, pure and simple, without any consideration of refining possibilities which were deemed unnecessary with the high-class raw material employed in the manufacture of such steel.

That this furnace has filled this demand, in capacities up to 4 tons and more, has in course of time been amply verified,

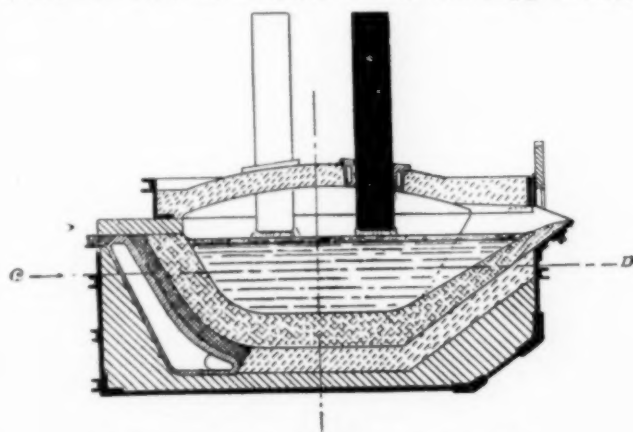


FIG. 1.—VERTICAL SECTION OF PARAGON FURNACE.

though the progress, the author regrets to state, has been very much retarded in England, chiefly on account of the power question, as "surplus" power is scarce in most steel plants and is considered more necessary for other purposes, though it has been repeatedly proved that a clear saving of from £1.10.0 to £2.0.0 (\$7.50 to \$10) per ton of steel, in the melting, can be effected with this furnace.

It is well known that very little refining as regards sulphur and phosphorus can be effected in the induction furnace on account of its small slag surface, and the comparatively low temperature of the slag (though degasifying is daily carried out in a large Kjellin furnace by a prominent Continental European firm, the Poldihütte A. G. in Bohemia).

The reason for this behavior is quite obvious, but a crude analogy may serve as an illustration. Imagine there are two cups of water on the table, one containing hot and the other cold water, and let us drop a lump of hard crystal sugar into both; as is well known, the sugar will dissolve much more rapidly in the hot water, also if some special salt is added, it may be found to dissolve quite readily in the hot, but not at all in the cold water.

The impurities in the steel may be considered to act precisely in the same way; they dissolve readily in a super-heated slag blanket of a temperature considerably higher than that of the steel, while a slag of the same or lower temperature than the steel may dissolve but little or none of the impurities.

As to the gases, another crude analogy may be taken for comparison. Take, for instance, a vessel containing soda-water, or any other liquid in which gases are occluded. If the flame of a burner is applied over the surface of the liquid, some of the gases may be made to escape somewhat quicker especially if the liquid is stirred, either by the action of the

heat or otherwise) but if the flame is applied from beneath the liquid, the degasifying is undoubtedly carried on much more quickly, as at the same time the lower portion of the liquid is rapidly brought to the surface and the gas expanded by the heat.

These considerations, among others, brought the author to work out the design known as the "Paragon" furnace, illustrations of which are herewith reproduced.

In this furnace, the bath is heated, as is easily seen (Figs. 1 and 2), both from the surface of the slag by means of suitably arranged arcs, and also at the same time from the sides and beneath the bath, by means of side plates of "second-class conductors," similar to those which have been used for some four to five years in the Roechling-Rodenhauser furnace. In this manner the metallurgist has it in his hands to apply the maximum heat exactly where he wishes to have it, since both circuits are made to be regulated at will. Thus, during the desulphurization and dephosphorization the slag is heated to a temperature higher than that of the steel, while during the period of degasifying, the bulk of the power is conveyed to the bath through the bottom and sides.

This design brings in other important improvements, viz.: It is quite easy to start the furnace from cold by means of the arcs, which obviates the necessity of filling in liquid charge as a means of starting the non-electrode furnaces. Furthermore, the electrode question in plain arc furnaces may in many cases become a serious one, as large electrodes for furnaces

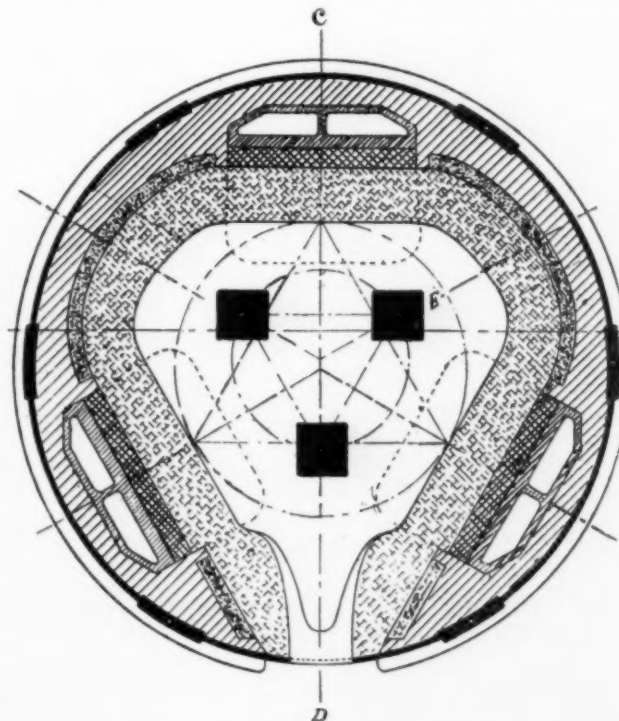


FIG. 2.—PLAN OF PARAGON FURNACE.

of greater capacities are exceedingly difficult to obtain, and always expensive, especially if the losses are considered.

Owing to the nature of the Paragon furnace, where only a smaller part of the power enters the furnace through the electrodes this drawback is considerably reduced. If, for instance, the upper limit of a plain arc furnace is, say 20 tons, on account of the difficulty in obtaining large enough electrodes, it will be found that this capacity can be doubled with the Paragon type, as only about half of the power is required to pass through the electrodes. It is true that with some designs of furnaces the electrodes may be coupled in parallel, but this is also possible in the Paragon, and this statement therefore holds good in this case also.

*A paper read before the Faraday Society on October 2, 1911.

The electrodes for a 30-ton three-phase Paragon furnace should have a cross section of 16-in. x 16-in., or 256 square inches, and in a 50-ton furnace the electrodes should be 24 in. x 24 in., or 576 square inches, which would still only give a maximum of 24.5 to 26 amp. per square inch, which, as experience has proved, is well within reasonable limits, both from an electrical and manufacturing point of view. This would correspond to plain arc furnaces having a capacity of only 12 to 14 tons and 18 to 22 tons. In fact, it may safely be said that as far as the electrodes are concerned any capacity possible with the plain arc furnace may be doubled with the Paragon design.

A further advantage in this design is the greater durability of the roof. It is well known that the roof is the part of an arc furnace which is most rapidly destroyed by the action of

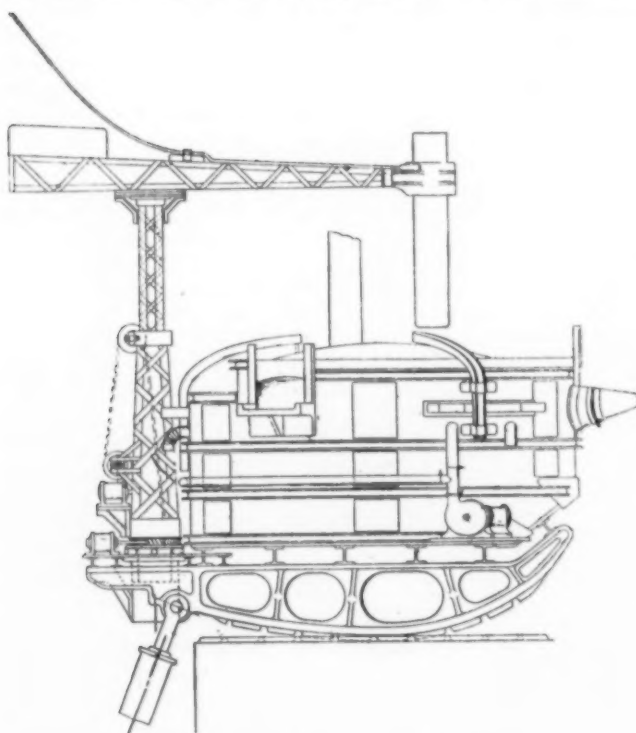


FIG. 3.—30-TON PARAGON FURNACE.

the hot gases. In the Paragon furnace the destructive action is minimized, as only a smaller part of the power is acting on the slag.

In cases where electrodes are more or less unobtainable, the same design may still be used with advantage, when gas firing is used as a substitute for the electrodes, while the electric power is applied only through the side plates. One may perhaps remark that in this case the gas firing alone will be sufficient, but it will be seen on closer consideration that the electric heating is still of great advantage, if the analogy with the liquid containing occluded gas, is remembered, which should be driven out.

Besides, the desulphurization and deoxidation require a considerable temperature, and in order to obtain this one has to blow in a sufficient quantity of air with the gas, which easily renders the flame slightly oxidizing, thus making the deoxidation of the steel difficult.

If electric heating is applied as described, the desired temperature can easily be obtained in the right direction by maintaining a reducing atmosphere, and a more thorough refining should be possible without incurring a higher cost.

Finally, the melting down of cold material with gas should be more economical in this particular case than with the electric furnace proper, since no very high temperature is required to liquefy low-grade iron; but it is the subsequent re-

fining, apart from the dephosphorisation, which demands the higher heat energy, and this can be applied in a more efficient way by means of the side plates conducting the electric power to the furnace.

These furnaces have been patented by the author and the Gröndal-Kjellin Co., Ltd.

A trial furnace is just being built in Germany and designed by Mr. Rodenhauser and the Roebling Iron and Steel Works, where the side plates only are provided for. Experiments have proved that sufficient power can be converted to the bath in this way to give the steel the temperature required. The construction of this furnace has progressed so far that the author hoped to give here some figures of the results obtained; certain delays in the delivery of the electrical parts have, however, frustrated this, but the results are expected to be ready for publication shortly.

It is expected that this design will enable the constructors to provide furnaces up to 50 tons capacity.

Having now broached the question of very large furnaces, in fact of the largest size possible under present conditions, the author wishes to mention a few words about a new furnace of the smallest size, suitable for laboratory purposes, etc.

It is a long-felt want of the steel-maker, to be able to carry out trial melts on a small scale, especially for alloys, before making up a proper charge. This is of great importance in the case of such alloy steels in which expensive ferroalloys are a prominent constituent.

The old method of making a trial in a pot in the crucible furnace is both time-robbing and very uncertain. This new furnace should be a real boon to the crucible steel maker, as it enables him to make melts from a few pounds in weight up to several hundred pounds in a most easy and controllable way.

The furnace is known as the "Helberger" furnace. It consists of an electrically heated ordinary plumbago crucible, lined in such a way as not to react upon the charge. Any form of electric current may be used, but alternating current of from 25 to 60 cycles is preferable.

The furnace and the crucible are open at the top, so that the contents can easily be watched and additions made. The temperature is very easily regulated by means of a handle; temperatures of 3000° C. have been obtained in these furnaces, which will even permit the melting of platinum and other very refractory metals.

The furnaces are made either to be tilted or else with detachable crucible for direct pouring; they are delivered complete, ready for use, at quite reasonable prices.

The melting cost compares very favorably with other methods for small capacities; a few figures will show this:

100 kg. steel requires	75 kilowatthours
100 kg. copper "	30 "
1 kg. platinum "	10 "

The following temperatures of the crucible were obtained:

In 8 minutes	1000° C. with	90 amperes,	104 volts.
" 15 "	1470 "	" 160 "	101 "
" 25 "	2150 "	" 165 "	100 "
" 45 "	2750 "	" 248 "	93 "

The temperature was still higher after the last reading, but the pyrometer used did not enable any higher readings.

From this it will be seen that any temperature required for trial purposes in steel-making may be easily obtainable.

With regard to accessories for steel-making, it may be stated that such ferroalloys as ferrotungsten are now being made in the electric furnace on a commercial scale in England by Electric Furnaces & Smelters, Ltd. in their new works at Luton. Three furnaces are in use alternatively, and alloys are produced in considerable quantities. Ferrotungsten, as low as 0.25% in carbon has been made, though 0.5% is more usual.

The power supply is 500 volts direct current which is being converted into alternating current, 25 cycles, 50 to 90 volts, by means of a motor generator.

London, England.

Iron and Steel Institute Meeting.

The autumn meeting of the Iron and Steel Institute was held in London on Oct. 5.

The Duke of Devonshire, President of the Institute, occupied the chair and was supported by two past-presidents, Sir Hugh Bell and Sir Robert Hadfield.

In his introductory address the president expressed great regret that it had been necessary to give up the original intention to hold this meeting in Italy. This was the more regrettable in view of the extensive preparations which had been made in Italy for the entertainment of the Institute. A resolution was adopted "that this meeting desires to place on record an expression of its most sincere regret that circumstances have arisen to deprive the Institute of the honor and privilege of holding the autumn meeting at Turin, under the auspices of its kindred organization, the Association for the Italian Metallurgical Industry."

There were a long list of papers of all of which printed advance copies were available. But only three of them came up for discussion. The others were read by title.

Transformation of Steel by Heat Treatment.

A paper by L. GRENET, of Paris, on "The transformations of steel within the limits of the temperatures employed in heat treatment" was read, in the absence of the author, by the secretary.

The chief conclusions of the author are as follows:

"The iron-carbon alloys partake of the general category of alloys, and no complications due to allotropic transformation of the element iron need be taken into consideration.

"It is quite easy to represent by simple diagrams the mode of influence of the special elements on the temperature of transformation on cooling.

"Quenching (rapid cooling from a temperature sufficiently high that the constituents of an alloy should not be the same as at the ordinary temperature) may, as compared with annealing (slow cooling), act as follows:—

"(a) By changing the nature of the constituents and, notably, by preventing wholly or partly the transformation on cooling. In this case, which is rare, so far, that is, as steels are concerned, the quenched state and the annealed state are physico-chemically different states, and their properties cannot be foreseen. It may, however, be said that in the few instances in which such quenchings are known the quenched state is often softer than the annealed state.

"(b) Quenching may also act by lowering the temperature of the transformation on cooling. In this, which is the most usual case, the quenched metal is always harder than the annealed metal.

"The following very general law, which satisfactorily groups the facts which have been observed as to steels of the most varied description, and appears to agree with what is known as to alloys, may be formulated. An alloy is the harder in proportion as the transformation which has given birth to its constituents at the time has taken place at a lower temperature, has been more rapid, and as the maximum temperature reached since the last transformation has been at a lower temperature and has been maintained for a shorter period of time.

"The pure metals are alloys in which one of the constituents is in the proportion of nil, and the law applies in this extreme instance, in which there is but one constituent (pure, or nearly pure, iron).

"When it is distinctly higher than the temperature of transformation, the action of the maximum temperature reached since the last transformation preponderates, and it is this which should serve as a guide in the treatment of steel.

"It would appear that the law enunciated above applies equally when the constituents for the time being result from a transformation which causes the metal to pass from the liquid state to the solid state (influence of the method of casting), as, for

example, in sand or in a chill; and when they result from a transformation of the metal to the solid state either before or after transformation (as in the quenching of steels).

"The phenomena relating to the hardening of a steel by quenching belong, therefore, to the exceedingly common category.

"So far as the real cause of hardening is concerned, it cannot be said to be known. We are thus led to the conclusion that hardening produced by quenching is due solely to the fineness of the structure. The comparison of the quenching of a solid solution to the quenching of a liquid solution (cast in chills) appears, therefore, to the author to be nearly perfect. In some instances the constituents of a metal vary with the rate of cooling (white or grey pig iron, austenitic steels); but, generally speaking, the constituents of the metal are independent of the rate of cooling, and the law connecting the hardness of the metal with the conditions under which the transformation has been effected applies. Annealing is liquation within a solid, and this liquation would appear to be limited only to the extent of the complete separation of the constituents. Exceedingly slow cooling isolates the constituents in steels or in alloys.

"Between the quenching of metal in a state of solid solution and the quenching of a liquid metal there are naturally enormous differences from the practical point of view. The cooling of a solid solution takes place without the intervention of a mold, and may be very rapid; the constituents separating from the liquid solution are more readily able to segregate by the intervention of the liquid solution not as yet decomposed than the constituents separating from the solid solution. The quenching of a solid solution is therefore more energetic than the quenching of a liquid solution.

"Further, while a liquid solution presents no discontinuity, a solid solution is divided into grains like the metals formed of a single constituent; the grains are larger in proportion as the metal has been heated longer and at a higher temperature. At the time of the formation of the 'cold stable state,' the texture of this state may depend partly on the texture on the 'hot stable state' which has preceded the state at the time being.

"What is important to remember as a guide in the treatment of alloys is the relation—*independent of any special conceptions as to the metal iron*—which connects the hardness of an alloy with the conditions under which the transformation on cooling is produced, and, in particular, with the maximum temperature to which the alloy has been raised since the setting at liberty of its constituents at the time being."

The paper was discussed at considerable length. Sir Robert Hadfield referred to the author's statement that if quenching acts by preventing wholly or partly the transformation in cooling, the quenched state is often softer than the annealed state. He could confirm this peculiar fact from observations on manganese steel.

Prof. I. O. Arnold said the paper showed that the rival schools of thought were now getting nearer together. He was especially glad that the author had come to the conclusion reached by the Sheffield school of metallographists long ago that the so-called martensite was not a constituent at all, but a structure.

Dr. J. E. Stead criticized the terminology employed by the author as differing in various respects from the usual one.

Dr. W. Rosenhain remarked that the author's opinion on the cause of hardening differed very much from that usually accepted in France or England. The usual view was that the hardening effect in quenching arose from the possible prevention of transformations which would occur on slow cooling. According to Grenet, however, hardness resulted from transformation occurring very rapidly and their products being present in extremely fine division. But this view would require experimental evidence to establish it.

Iron-Manganese Alloys.

Prof. J. O. ARNOLD, of the University of Sheffield, then presented a paper by himself and F. K. KNOWLES on "the mechanical influence of carbon on alloys of iron and manganese."

Arnold and Read had formerly expressed the opinion that "the influence of carbon is very considerable, but is by no means the dominating factor, which without doubt is the manganese."

The object of the present paper is to prove the truth of the foregoing enunciation.

Two series of steels were made, in both of which the manganese ranged from 1 to 20 per cent. In one series, however, the carbon was about 0.85 per cent, while in the other it was kept under 0.1 per cent. The results are given of tests of the turning properties, static mechanical (tensile) tests, and dynamical stress-strain tests (alternations induced). Microphotographs of the structure of four typical nearly carbonless alloys are given.

Sir Robert Hadfield, in discussing the paper, gave hardness figures of the manganese steels, obtained by the Brinel method of testing hardness, which he said should find much wider applications, as very valuable practical conclusions could be drawn from it.

Dr. J. E. Stead urged that the investigation should be extended to other properties of the manganese steels.

Temperature Influences on Carbon and Iron.

Mr. E. ADAMSON, of Sheffield, in his paper on "temperature influences on carbon and iron" endeavored to show that the determination of silicon alone is not sufficient in the chemical grading of iron.

The author gives facts to show that carbon is precipitated as free carbon virtually in the absence of silicon, and in the author's opinion this precipitation in the case of the manufacture of pig iron, primarily depends upon the original temperature to which the iron is raised in the blast furnace.

The higher the temperature in the blast furnace the higher the carbon in the product—within limits—for it is well known that cold-blast iron is lower in total carbon than hot-blast iron, although the full force of this fact has perhaps been overlooked. What this amounts to is that the temperature of the furnace decides the fracture of grey iron as well as of the lower grades, regardless of the contents of silicon, sulphur, phosphorus and manganese.

The author distinguishes between three different varieties of free carbons and discusses under what conditions they appear. Analyses are given to show that under no conditions can it be said that the percentage of silicon controls the percentage of combined carbon or the graphite or the ratio of one form to the other. Under precisely the same conditions of temperature and time, additions of silicon may show fairly regular variations in the carbons, but under the same constant conditions phosphorus and manganese would do the same.

"The facts stated in this paper are of vital importance in considering the question of the grading of pig iron, and to a large extent explain the objections of many in this country to complete change in the methods used at present to one of grading by practically silicon contents only. The author has shown on several occasions that it is not possible to regulate tests by silicon only, and has no hesitation in saying that the fracture indication is quite as satisfactory as varying the silicon, if the user is fully conversant with fracture indications.

"Continual experience in the use of pig irons in the foundry shows that the same tests can be obtained within very wide ranges of silicon contents. Of course, if it is a question of commerce, there should be no objections to selling irons by silicon contents if this is demanded, and full responsibility is taken by the purchaser; but when it becomes a question as to whether such a method of grading is the best, so as to be of the greatest use to the greatest number, the reply must be an emphatic negative, for it would appear that carbon and its

various conditions, due, as shown, to influences entirely outside of analyses, is of primary importance."

In the discussion Dr. J. E. Stead said that the paper was an attack upon analysis and also an attack upon those who depended upon the determination of silicon for pig iron. He thought that all those who had anything to do with foundry work understood thoroughly that the determination of silicon was most important.

Mr. Adamson replied that he was not attacking the chemist, who was a very valuable man as he enabled the iron founder to use grades of iron which he would otherwise not be able to employ. But he attacked grading by analysis pure and simple, and this meant in fact grading by the silicon content.

The other papers were read by title. Nine of them were from the pens of Italian engineers and gave very interesting data on metallurgy in Italy. On account of limitations of space only the two papers on case-hardening are abstracted in this issue. Abstracts of the other papers are reserved for our next issue.

Case Hardening of Steel by Carbon Monoxide and Carbon

Dr. F. GIOLITTI's paper gives first a summary of the facts established in recent years by the author in co-operation with others on the theory of case-hardening (the original papers having appeared in *Gazzetta Chim. Ital.*, vol. 38, No. 2, 1908; vol. 39, No. 2, 1909; vol. 40, No. 1, 1910; *Rendiconti della Soc. Chim. di Roma*, vol. 6, No. 17, 1908; *Atti della Reali Accad. delle Scienze di Torino*, vol. 45, 1910; vol. 46, 1911; *Rassegna Miner., Metall. e Chim.*, vol. 34, No. 18, 1911, and September, 1911; *La Metall. Ital.*, August, 1911, and September, 1911.) These principal facts are as follows:

Where case-hardening is carried out with solid cementing agents having a carbon base, the carburizing effect of the free carbon on the iron, the materials being in simple contact and without the intervention of gaseous carbon compounds, is exceedingly weak, and in any case is entirely negligible in industrial practice.

Where case-hardening is effected with the solid case-hardening agents ordinarily employed in the industry, the specific effect of the nitrogen is very weak. Only with the cementing agents containing a high proportion of the cyanogen compounds (alkaline cyanides, ferrocyanides, etc.), does the direct action of volatile nitrogen compounds have any marked effect.

As compared with the cementing effect of the solid agents ordinarily employed in the industry, the specific direct carburizing effect of carbon monoxide preponderates enormously over every other carburizing effect.

Pure carbon monoxide carburizes iron at all temperatures within the range (700 deg. to 1300 deg. C.), at which the process of case-hardening can be performed by means of any other medium whatever. Moreover, the rate of case-hardening (by which is understood the depth of carburization which can be obtained in a given time) when working under suitable conditions is greatest when carbon monoxide, or a mixture in which the carbon monoxide can efficaciously exercise its specific carburizing effect, is used as the agent.

This specific carburizing effect exerted by the carbon monoxide on the iron at high temperature is due to a series of chemical reactions, the course and state of equilibrium of which has been actually observed with precision. Moreover, the conditions of equilibrium of the systems in which these reactions take place are in general comprised within the ranges of temperature and pressures ordinarily employed in practice. It is, therefore, possible to obtain with certainty a predetermined result using case-hardening agents whose activity is due, if not exclusively at least very largely, to the specific carburizing action of the carbon monoxide. More particularly is it possible to obtain with such agents carburized zones in which the concentration of the carbon does not exceed a predetermined maximum limit and varies in a well-defined degree toward the inside of the carburized zone. Such definite results, variable at all within sufficiently wide limits, are obtained by varying, in accordance

with fixed rules, the temperature at which the case-hardening is performed, the pressure of the carburizing gas, and the amount of carbon monoxide which, in a given time, comes in contact with the unit surface of the steel to be carburized.

The results obtained by the use of carbon monoxide as an agent vary regularly, other conditions being equal, as the chemical composition of the steel to be carburized is varied.

It is possible to vary regularly and within fairly wide limits the characteristics of the desired product by subjecting the steel to the action of substances along with carbon monoxide which are capable of modifying the conditions of equilibrium of the chemical systems under which the reactions due to the specific carburizing effect of the carbon monoxide are completed. Such substances may be gases, such as hydrocarbons or nitrogen, or they may be solids such as carbon in various forms, and their actions can proceed simultaneously with that of the carbon monoxide throughout the whole cementation period or during a portion only of it.

In particular, by means of the agents, the activity of which is due to the specific carburizing effect of carbon monoxide, it is possible to obtain with ease and certainty—whatever kind of steel is being operated upon—soft case-hardenings and graduated case-hardenings; that is to say, carburized zones in which the concentration of the carbon without being excessive in the outer layers, diminishes slowly and with regularity in the succeeding deeper layers. This is the essential condition for the avoidance of the dangerous phenomena of brittleness and peeling, which defects manifest themselves so frequently in steel pieces case-hardened by the processes ordinarily used in the industry.

The chemical reactions produced by agents in which cyanogen is the active element, are at present but imperfectly understood, particularly as regards their condition of equilibrium upon which depends the concentration of carbon in the carburized zones. It is, however, certain in the conditions under which case-hardening should be performed in practice that the conditions of equilibrium just now alluded to should closely correspond to the strength of the very high concentration of carbon passing into solution in the iron. Thus it happens that the cyanides, ferro-cyanides, and other derivatives of cyanogen, if used alone as case-hardening agents, always give rise to too rapid (energetic) a case-hardening. That is, carburized zones are produced in which the concentration of the carbon is excessively high in the outer layer up to a certain depth and is then suddenly lowered in the succeeding layers. Zones of that type produce brittleness and peeling.

Further, gaseous or volatile hydrocarbons, when used alone as agents, also give rise to too rapid case-hardening, the causes being identical with those referred to in the case of cyanogen and its compounds.

In the light of the facts recited in the foregoing, the great advantage of the use of agents, the activity of which is due, if not exclusively, at least principally, to the specific carburizing action of carbon monoxide, is clear. In order to obtain the best results with such an agent, it is necessary to satisfy the following fundamental conditions:

1. The chemical composition of the agent should be absolutely definite, and should be accurately known.

2. The compounds should be as simple as possible.

3. The reactions which take place during the case-hardening process between the various constituents of the agent and those of the steel should be simple, and should proceed rapidly—under the conditions most easily obtaining in practice—to a well-defined state of equilibrium corresponding to definite concentrations of carbon in the carburized zones.

If pure carbon monoxide alone is used, the concentrations of carbon in the carburized zones are generally too low. It is, therefore, preferable to use along with the carbon monoxide small quantities of hydrocarbons of known composition, or to use along with the carbon monoxide solid carbon in a properly divided state. It is the latter method—the combination of car-

bon monoxide with free carbon—which is advocated by the author.

Among the numerous advantages claimed for this method by the author are great speed of penetration of the carburized zone; great uniformity in the distribution of carbon in the carburized zone; possibility and ease of regulation of carbon concentration so as to get maximum hardness with minimum brittleness; absolute security against the introduction of foreign substances into the steel; a minimum of deformation and change of volume of the steel pieces.

While the process may be carried out in the ordinary horizontal muffle generally used for case-hardening, the following special furnace offers decided advantages.

As shown in the illustration on p. 600, it consists of cylindrical muffles arranged vertically and heated by producer gas. It is a double-muffle furnace provided with regenerators, of the kind constructed by Messrs. C. M. Stein, of Paris, for the Sampierdarena Engineering Works of Messrs. G. Ansaldo, Armstrong & Co.

The parts of the apparatus particularly applied to the case-hardening process are represented in the illustration which shows a section through the center of the vertical muffles of the furnace with the accessory apparatus completely mounted.

Inside each of the muffles of refractory material is placed a cylindrical retort of mild steel, A, the external diameter of which is about 10 to 20 millimeters less than the internal diameter of the muffle. Mannesmann seamless tubes serve very well for making the retorts, as they are easily obtainable in stock sizes up to 35 centimeters diameter. For retorts of larger diameter, as in the case of the furnace here represented, tubes welded by the autogenous process are used. The retort is supported on a bottom flange attached to a frame fixed to the brickwork of the furnace. The arrangement which serves to hold the retort, whether it is being supported by the bottom flange or by the upper flange, H, intended to receive the cover, is such as to enable one retort to be replaced by another in a few minutes. To the bottom flange is attached a cast-iron funnel B, closed at the bottom by a non-return valve Q.

The steel nozzle C, admitting the carburizing gas, passes through an opening in the side of the funnel, provided with a good-sized boss. The gas is introduced through the screwed tube D, and through the nozzle C. This latter is connected to the hollow cast-steel dish E, intended to support the pieces to be case-hardened by means of the disc F, consisting of a steel plate covered with refractory material.

The carbon monoxide, after passing through the hole bored axially in the nozzle C, is delivered into a small distributing device placed inside the dish E, from which it passes to the case-hardening chamber G, through a number of holes provided in the disc F. During case-hardening the dish E is supported upon a series of projections on the same lower flange to which the funnel B is attached. This ring has thus to sustain the entire weight of the dish E, the disc F and any articles to be case-hardened which may be resting directly or indirectly on the disc. The section shows exactly the position of the various parts during case-hardening.

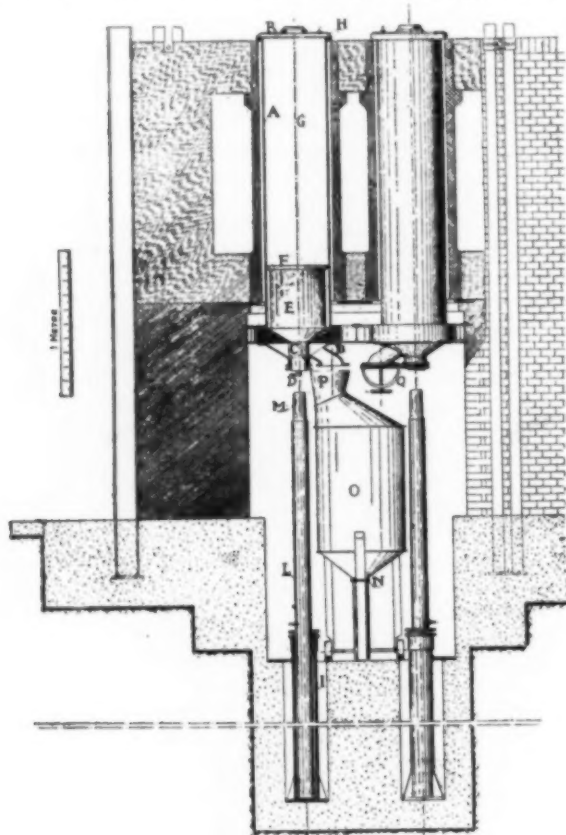
In the case of spur wheels, the maximum diameter of which might be 100 to 150 mm less than the internal diameter of the retort, the first thing to be done is to unscrew the tube D; then by means of the plunger L, actuated by the hydraulic cylinder I, the whole of the parts C, E and F are lifted together, the blunt conical end of the plunger M engaging with these as it rises. The plunger LM is cooled internally with water circulation, so as to avoid the heating of it in case it should accidentally remain raised inside the retort for too long a period.

When the piston L has reached the end of its stroke the upper surface of the disc F being now only about 30 cm below the cover of the retort; the cover is removed, and the wheels to be case-hardened are piled horizontally one on top of the other upon the disc F. As the wheels are inserted

one by one, the apparatus carrying the weighed portions C, E and F gradually sinks, and the charge is complete when the apparatus has reached the lowest position indicated in the figure. In that position the last wheel charged should be at least 30 cm below the upper edge of the retort.

Generally, the articles to be case-hardened are charged hot at a temperature of about 800° to 900°C, this being a matter of prime importance, as it affects most favorably both the quality of the product and the economy and regularity of the operation.

For the preliminary heating of the pieces a small coal-fired muffle will serve, since the uniformity of temperature which



CASE-HARDENING FURNACE.

can be obtained in such a furnace is quite sufficient for the purpose. If, however, the case-hardening furnace is not being worked to its fullest capacity, one of the vertical muffles may be used for the preliminary heating of the articles, while the case-hardening of the charge is proceeding in the other. In this case the best method of working the furnace is to use the muffles alternately, the one for case-hardening and the other for preheating the pieces.

As soon as the articles to be case-hardened have been inserted as described, the cover is placed on the retort and the bottom discharge pipe N, leading from the iron box O, is passed through the central opening in the cover. This box is full of the granular carbon still hot (generally at about 900°C.) which was discharged from the retort after use during the last case-hardening operation just concluded. The box is easily swung above the muffles, being suspended by a tackle from a swinging arm.

The outer walls of the box, notwithstanding that its contents of granular carbon are at a temperature of nearly 1000°C., do not become heated above 200° to 250°C. during the time required for the loading and unloading of the retort, even when working under the most unfavorable conditions. This is due to the small thermic capacity and the very low conductivity of the mass of granular carbon, which, although at a high temperature, can only impart to the walls of the box containing it a small pro-

portion of its heat in a unit of time. The heat is quickly dissipated simply by the cooling action of the air currents passing around it.

The box O having been adjusted as indicated, the butterfly valve which closes the low end of the opening N is gradually opened, and in a few seconds the whole of the free space in the retort surrounding the pieces is filled with glowing granular carbon. The hot granular carbon employed by the author for case-hardening with his mixed medium constitutes a mass which, particularly at high temperatures, is endowed with a mobility comparable to that of a liquid. On this account it is able to penetrate simply by its own weight into all crevices among the articles to be case-hardened, and between these and the walls of the retort. The filling up may be further assisted by means of open rods introduced through holes in the cover and manipulated by the operator who stands on the furnace.

When the granular carbon has been filled in up to a level of 2 cm or 3 cm below the upper edge of the retort, the butterfly valve of the tube N is closed, the box O is lifted, the central opening of the cover is closed, the plunger L is lowered right down and the tube D is screwed into the nozzle C, through which the carbon monoxide or dioxide, pure or mixed with air, is admitted gradually to the distributor E and then into the retort. In order to regulate the current of carbon monoxide, an ordinary gas meter of the dry type is used. The case-hardening then proceeds without further trouble, the only thing that is necessary to be watched and regulated being the current of gas and the temperature.

For the observation of the temperature, an operation which, with the type of furnace described and when adjusted properly, does not often require to be made, an ordinary thermoelectric pyrometer is used by passing it through any one of the holes in the cover from time to time and inserting it into various parts of the mass of granular carbon. The temperature should be maintained quite constantly at a fixed point between 900° and 1100° C., chosen in accordance with the rules laid down by the author in the reports previously referred to.

The gas issuing from the retort through the pipe fixed in the cover can be collected in a gasometer, and after suitable regeneration it will serve for further heats. The necessary length of the heating period can be determined with the greatest precision, and is dependent on the result which it is required to obtain. When the heating has proceeded long enough, the current of carbon monoxide is shut off, the tube D is removed and the discharge of the retort is undertaken.

Where results of a very special nature are not aimed at, the discharging operation begins by withdrawing the whole of the granular carbon at once, and removing it through the non-return valve Q in the box O, now placed underneath.

Where, however, exceptionally graduated case-hardening is required, the specific action of the carbon monoxide is isolated during the last phase of the case-hardening from the direct action of the granular carbon. In that case the box O is placed so that the neck of the funnel B is inserted into the open end of the tube P, the box for this purpose being placed in the chamber below the muffle, where it can be maneuvered on a small bogie.

The non-return valve is then opened gradually and a quantity of the granulated carbon is drawn out from the retort. Only those parts of the case-hardened pieces are uncovered in which a well-graduated case-hardening is required. Sufficient solid carbon is retained in the retort to ensure the desired chemical equilibrium of the gas. The remainder of the carbon is removed from the retort after the second phase of the case-hardening with isolated carbon monoxide has proceeded long enough to attain the desired results.

In either case as soon as the whole of the granular carbon

¹The carbon contained in the retort falls out from below, passing first through a series of holes in the bottom flange upon which the apparatus C, E, F rest. It was, however, not possible to show all these holes in the accompanying sectional view of the furnace.

has been evacuated from the retort, the cover of the latter is lifted. In performing that operation the operator must be careful to remain at some distance from the upper open end, because the air which enters forms, with the carbon monoxide contained in it, a mixture which explodes suddenly at the high temperature of the retort. The explosion, which in any case is light and not at all dangerous, may be avoided by rapidly scouring the muffle with a little carbon dioxide after the removal of all the solid carbon. Care must be exercised, however, to perform this operation as quickly as possible, in order to prevent the carbon dioxide causing a superficial decarburization of the case-hardened pieces.

When the pieces rest direct on the disc F, as in the present supposed case, they may be lifted by means of the appliances C, E, F, up to the open top of the muffle without first discharging all the granular carbon from the retort. In that manner the light explosion just referred to is avoided, but the picking up of the pieces for quenching is made more difficult, and a considerable quantity of the solid carbon is apt to get burnt in the retort.

At all events, whether the carbon is removed from the retort or not, as soon as the cover R is removed the plunger L is raised until its conical head beds itself in the recess provided for it in the part C. The hydraulic ram then continues to move slowly, gently lifting the weight-supports, C, E, F, with the case-hardened pieces resting on them. It only remains then for the operator standing above the furnace near the open muffle to pick up the case-hardened pieces one by one as they come within reach and quench them in the tank, also placed above the furnace (in case double quenching is desired), or bury them in hot ashes. When all the pieces have been removed, the furnace is ready for a fresh charge.

The method of loading the pieces to be case-hardened into the retort varies greatly according to their shape or size. Pieces of cylindrical shape can be piled on one another as in the foregoing example. If the pieces are of small diameter, such as pinions, cutters, cups and rings for ball bearings, several columns can be built up for one heat; but in that case it is desirable to keep the pieces in position by threading them on iron rods, assuming that the external surfaces only require to be case-hardened.

For cylindrical pieces to be case-hardened internally, such as rings for ball bearings, the rods are placed outside the columns. Pieces of great length or of irregular shape have also been suitably supported in the retort. Finally, very small pieces are placed in cages formed of iron framework held together with iron wire netting. Such pieces can also be laid free on successive layers of granular carbon.

The operation of charging the retort may vary from $4\frac{1}{2}$ to 6 minutes and may be increased to 10 or 12 minutes. The time required for the complete discharge of the retort is $3\frac{1}{2}$ to $5\frac{1}{2}$ minutes, though it may extend to 13 or 14 minutes.

Further, in the case of the process described and in almost all cases occurring in the practical production of case-hardened pieces for machine construction, in which the carburized zone is seldom required to exceed 2 mm in thickness, the period of case-hardening never exceeds 2 hours, and may even be reduced to one hour, the time being reckoned from the moment of completing the charging operation to that at which discharging begins. It, therefore, includes the time necessary for bringing the temperature up to working point, which requires not more than 10 minutes, owing to the fact that both the solid carbon and the pieces are already hot when charged.

The foregoing particulars show that in practice, under ordinary working conditions, the time for a complete heat is less than $2\frac{1}{2}$ hours, and since the weight of the charge, which retorts of the size shown in the accompanying sketch can conveniently hold, varies from a minimum of 100 kg in exceptionally unfavorable cases to a maximum of over 500 kg, it will be seen that the productive capacity of each retort ranges from a minimum of 1 ton to a maximum of 5 tons

of case-hardened steel pieces during a working day of 24 hours.

The chief advantages of the furnace described over horizontal muffles are greater speed of working, greater simplicity of operating, and increased uniformity of the treatment of all the pieces forming the charge in the muffle.

The author finally gives two examples which show what can be done in practice with this process, and especially how the distribution of the carbon in the carburized zones can be graduated by isolating the action of the carbon monoxide during a part of the carburizing period.

The following data refer to case-hardenings of medium depth (from 5 mm to 10 mm) in which the application of the process in question presents the greatest technical interest, particularly in the case of very deep case-hardening. The tables give the contents of carbon in successive layers of a thickness of 0.5 mm, obtained by case-hardening with the mixed agent, followed by exposure to the action of carbon monoxide isolated in the manner previously described.

First Case.—The material used was an ordinary mild steel containing 0.12 per cent C, 0.06 Si, 0.47 Mn, 0.02 S, 0.03 P.

The figures in the column marked *a* give the amount of carbon in the different layers after case-hardening for ten hours at 1100°C . with the mixed agent; *b* gives the amount of carbon after reheating the same carburized zone for five hours at 1100°C . in isolated carbon monoxide; *c* gives the amount of carbon after reheating the same zone for another five hours (altogether ten hours of reheating) at 1100°C . in isolated carbon monoxide.

No. of range of single layers analyzed.	Distance of layer analyzed from surface.	Amount of carbon.		
		<i>a</i>	<i>b</i>	<i>c</i>
1	0.5 mm	1.17	0.85	0.70
5	2.5	0.81	0.78	0.67
10	5.0	0.34	0.46	0.53
15	7.5	(0.12)	0.25	0.39

Second Case.—The material used was a chromium-nickel steel of the kind ordinarily used for Krupp armor-plates. On this account the data here given relative to case-hardening at moderate depth are of peculiar interest. The steel contained 0.33 per cent C, 0.06 Si, 1.15 Mn, 0.02 S, 0.015 P, 1.50 Cr, 3.17 Ni.

The results after each single phase of treatment were as follows. The figures under *a* give the amount of carbon after case-hardening for ten hours at 1100°C . with the mixed agent, *b* the amount of carbon after reheating the same carburized zone for five hours at 1100°C . in isolated carbon monoxide, *c* the amount of carbon after reheating the same zones for a further five hours (total reheating period ten hours) at 1100°C . in isolated carbon monoxide.

No. of range of single layers analyzed.	Distance of layer analyzed from surface.	Amount of carbon.		
		<i>a</i>	<i>b</i>	<i>c</i>
1	0.5 mm	1.16	0.80	0.86
5	2.5	0.81	0.77	0.87
10	5.0	0.50	0.58	0.75
15	7.5	(0.33)	0.45	0.59

The author concludes that his process permits to obtain carburized zones in which are eliminated the principal causes of brittleness and peeling which always occur, in a greater or less degree, in steels case-hardened by the ordinary process. The principal causes of these defects are excessive concentration of carbon in superficial zones of the case-hardened pieces, and the rapid diminution of the carbon in passing to the deeper layers. It is also worthy of note that this rapid diminution contributes in a high degree to render more intense the phenomenon of liquation of the cementite or of the corresponding complex carbides of the various special steels, and of the ferrite. These phenomena in their turn are productive of the sudden local variations of the concentration of carbon in the carburized zones to which is due the peeling off of the carburized and quenched zones.

Finally the ability to alter the form of the curves of case-hardening, as shown in the two examples, suggests that case-hardening can also be successfully applied to those special steels on the exterior of which the ordinary process of case-hardening yields carburized zones, which are practically useless on account of their excessive brittleness.

Case-Hardening by Means of Compressed Gases.

A paper by F. GIOLITTI and F. CARNEVALI, both of Turin, Italy, deals with the effect of gaseous pressure in case-hardening by means of compressed gases, especially by the method (described in the preceding paper in detail) employing the "mixed agent" of carbon monoxide gas and free carbon.

The authors have formerly found that the characteristics of cementation zones vary in notable fashion concurrently with the variation in pressure of the carburizing gas, in the sense that an increase of pressure produces (all other conditions remaining unchanged) an increase in the depth of the cementation zone obtained within a given time and an increased concentration of carbon within the zone itself.

The authors have investigated this phenomenon quite carefully. The arrangement of their experiments is described in detail in the paper and the results of the treatment of six different steels are given. The case-hardening was accomplished by means of a mass of wood charcoal traversed by a slow current of carbon dioxide. This gas supplies with great rapidity and without any possible excess of carbon monoxide, a mixture of carbon dioxide and carbon monoxide of a concentration exactly corresponding to the equilibrium with free carbon, under the conditions of temperature and pressure employed in the operation.

The tests tabulated in the paper show clearly that an increase in pressure of the carburizing gas increases the depth of the case-hardening and the concentration of the carbon in the carburized zone as stated before.

A secondary effect observed is that an increase in the velocity of the current of carbon dioxide tends to cause a diminution in the intensity of case-hardening to such an extent as to eliminate the effects of increase of pressure. This is due to the fact that, on account of the high speed of the carbon dioxide, the carbon monoxide gas does not reach a concentration corresponding to equilibrium.

A third effect which occurs frequently in the course of case-hardening operations carried out under high pressure by means of a case-hardening mixture based on carbon monoxide is the superficial oxidation of the case-hardened steel. This effect is made the subject of a long discussion of the theory of the process.

Finally the authors remark that an understanding of the theory of the process permits to draw valuable practical conclusions.

One example is the fact that the case-hardening of steels with rather high percentages of chromium (more than 4 or 5 per cent) with "mixed cement" at a temperature not higher than 1000 deg. C. (a limit which in many cases cannot be exceeded) is accompanied by considerable oxidation of the metal, even when the process is conducted below ordinary atmospheric pressure.

Now, the results of the tests of the authors suggest immediately a method of avoiding this oxidation; it will suffice, indeed, to reduce the pressure of the carburizing gaseous mixture ($\text{CO} + \text{CO}_2$) below the minimum pressure of oxidation characteristic of the particular metal at the temperature at which it is being case-hardened. This can be in many cases easily accomplished by "diluting" with air the carbon dioxide which is circulating through the mass of granular carbon. In this way the carburizing gaseous mixture acts under partly reduced pressure, since it is "diluted" by the nitrogen of the atmosphere.

Experience has abundantly confirmed the efficacy of this method.

Abstracts of the other papers presented at the meeting will be given in our next issue.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

The Institute of Metals.

The autumn meeting of the Institute of Metals took place on the 20th, 21st and 22nd of September at Newcastle-on-Tyne. It is noteworthy that the whole of the papers dealt with non-ferrous metals and alloys, and that the subject of corrosion in connection with such metals and alloys received a considerable share of attention. Abstracts of the papers presented were given in the October issue, page 515. In the following we give an account of the discussions:

Non-Ferrous Metals in Railway Practices.

Mr. GEORGE HUGHES, of the Lancashire & Yorkshire Railway, read a paper on "Non-ferrous Metals in Railway Work" (abstracted in the October issue, p. 516.) The paper extended only to the use of non-ferrous metals in the locomotive and rolling-stock departments, and was peculiarly valuable on account of the author's description of the means of overcoming difficulties liable to occur in course of manufacture. The chief metals dealt with were copper and copper alloys and aluminium.

The discussion was opened by Mr. A. V. Hussey who said that the successful soldering of aluminium yet remained to be achieved; but the autogenous welding process was being largely used with satisfactory results.

Dr. G. H. Desch said that the statement that copper containing 0.3 per cent of arsenic was preferable for fire-box stays was not in accordance with usual experience.

Mr. H. Woore remarked that with phosphor-bronze slide valves running on cast-iron faces, he had found the wear to vary very widely, and he was inclined to ascribe this variation to differences in the cast iron.

Prof. Huntington, with regard to the use of bronze slide valves working with super-heated steam, said that the Lancashire and Yorkshire Railway seemed to have had very good fortune, but as the employment of super-heated steam in locomotives increased it was certain that the discussion of the subject would be revived.

Mr. L. Boeddecker desired to point out that the author had neglected the good points of copper-nickel alloys. The alloy containing 2 per cent of nickel possessed the remarkable property of increasing its tensile strength, after long annealing, from 15.2 to 16.1 tons; and there was a further improvement to 17.1 tons per square inch after five hours' heating in the fire-box of a locomotive.

Prof. H. Louis said that, although the paper compared copper with steel fire-box plates, no comparative figures were given for best Yorkshire iron, and he thought figures for that material were desirable.

Prof. Carpenter pointed out that the specification for copper boiler tubes especially directed that the ends should not be annealed; but this was quite contrary to the proposition of Dr. Beilby that all copper which was to be used for super-heating should be annealed at 300° to 350°C.

Sir Gerard Muntz said it was gratifying to know that copper nickel alloys were being experimented with, and it was probable that they would displace the alloys of copper with arsenic. The alloy containing 2 per cent of nickel was stated to retain a tensile strength of 14 tons up to a temperature of 700°C. Pure copper, under the same conditions, gave only six tons; and the usual arsenical copper about ten tons per square inch. There was one point which frequently appeared to be lost sight of in the practical use of copper, and that was the very great influence of the oxygen content on the qualities of the copper.

Hard-Drawn Copper.

A paper on "The Mechanical Qualities of Hard-drawn Copper" was contributed by Mr. D. R. PYE, who directed attention to the want of any satisfactory standard specification for

the properties of hard-drawn copper and the consequent variation of metals so described when obtained from different manufacturers (in October issue, p. 516). He suggested as a satisfactory definition of hard-drawn copper, "Hard-drawn copper, when in the form of circular wires, should have a tensile strength not less than that given by the formula $T = 30 - 20D$, and an extension per cent on a marked 10 in. length including the point of fracture, of not less than that given by the formula $e = 5D$, where T = tensile strength in tons per square inch of original section, e = extension per cent, and D = diameter of the circular rod in inches."

Mr. A. P. Trotter said that further tests were clearly necessary, as tests of material furnished by one manufacturer only did not constitute a satisfactory basis for action; and manufacturers were chary of making their results public property. They would not give any description of their processes; and users of hard-drawn copper had to be content with what they could get.

Sir Gerard Muntz said it was curious that the amount of work put upon the metals was not taken into account in drawing up specifications. It was his experience that the same specifications were received repeatedly both for 16-gauge wire and for $\frac{1}{2}$ -in. wire, and of course the differences of diameter should be allowed for.

Conductivity and Constitution of Alloys.

Dr. W. M. GUERTLER read a paper on "The Electrical Conductivity and Constitution of Alloys" in which he discussed the determination of the constitution of alloys from their conductivity (abstracted in the October issue, p. 517).

Dr. W. Rosenhain opened the discussion and said that the paper was useful in that it directed attention to the limitations of the methods habitually employed by metallurgists in ascertaining the composition of alloys, especially the limitations of thermal analyses. All would welcome the additional method of investigation, but that method also had its limitations. He did not intend to depreciate the value of electrical methods, and thought that metallurgists had unduly neglected those methods in giving preference to thermal results and micro-structure.

Dr. C. H. Desch considered that the author had furnished distinct evidence of the existence of inter-metallic compounds in solid solution; and that had recently been extended to the region above the liquidus curve. There was thus proof of the existence of compounds in a liquid state such as had hardly been obtained before with aqueous solutions.

Prof. T. Turner said he was glad that the author had indicated the inutility of expecting to get good conductance in an alloy. Metallurgists seemed to be unable to solve the problem; and it remained to work for the improvement of the mechanical properties of the alloy to a greater extent than the conductivity decreased.

Copper-Tin Alloys.

"Volume Changes in the Alloys of Copper with Tin" was the title of a paper by Mr. J. L. HAUGHTON and Prof. T. TURNER, from which the following conclusions were drawn: (1) the expansion during solidification is closely related to the crystallization interval, except in two regions, but the nature of the relationship is not clear; (2) allotropic changes, during and after solidification, cause variations in the coefficient of expansion; (3) accurate determination of the densities of copper-rich bars on original castings is rendered impossible by blow-holes; and (4) the hardness curve does not fit in with Farnakow and Shemtschusny's theory, probably owing to the metastable condition in which the bars exist. (See also October issue, p. 515.)

In the course of a brief discussion Dr. Rosenhain remarked that the method described in the paper was suitable for rough analysis, but not for ascertaining the composition of the complicated alloys of the copper-zinc series.

Dr. Desch said that the effect of crystal thrust complicated the interpretation of the results.

Prof. T. Turner, in reply, claimed that the experimental methods adopted were duly adequate; and that the volume changes measured corresponded exactly with those obtained in practice. The results arrived at had moreover been corroborated by the researches of Keep and Wüst.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the eight months ended on the 31st August, put the value of imports of iron and steel, including manufactures, at £7,369,674, and exports at £28,605,040, the respective increases being £1,615,364 and £307,944 as compared with the corresponding period of last year. The imports of other metals, including manufactures, were £18,160,533, and exports £7,071,749; increases of £2,701,540 and £397,887. Imports of electrical goods reached £887,215, with exports of £1,781,520, a decrease in imports of £89,404 and a decrease in exports of £821,516. The value of imported machinery was £3,984,669, and of exports £20,417,462; increases of £986,015 and £1,352,224 respectively. Imports of new ships amounted to £56,796 and exports to £3,703,481, an increase in imports of £39,063 and a decrease in exports of £3,349,499. Taking the month of August only, during which the strikes had a considerable effect on trade, all the exports except those of machinery were materially lower than in August, 1910. Iron and steel fell by £177,452; other metals dropped £75,735; electrical goods fell £400,960; while the value of ships declined by £1,713,224. In imports for August, iron and steel show an increase of £104,314; other metals rose by £3,763, and machinery by £29,339, but electrical goods dropped £43,378 and new ships £1,793.

Electric Steel.

An article entitled "Electric Steel: Its Position and Requirements," which appeared in the *Times* of the 9th August, has given rise to a prolonged and decidedly interesting controversy in the columns of that newspaper, from which it would appear that the views of the writer of the article are widely shared by steel users in England, and to some extent in other countries too.

One correspondent points out that a power consumption of 540 kw per ton in a two and a half ton Héroult furnace, would mean the extraordinarily high furnace efficiency of 79.25, but would only effect the melting of scrap, and not the making of steel or the killing of the metal. Another questions the lasting or fatigue properties of electric steel giving the same analysis and having the same tensile strength and elongation as basic hearth steel. But, without going into the various points discussed, it may be stated that unless and until much greater uniformity of quality and properties than are at present obtainable in electric steel can be secured there is little prospect of the new product making much headway here, especially among tool makers.

A flood of light is thrown on the subject by the remark of a certain German file manufacturer, who after using electric steel for a time handed an order to a Sheffield steel traveller, saying "Thank goodness I am going to have a steel again from which I can get more than two files alike out of a bar."

On the other hand, English steel-makers recognize the possibilities of electric steel; and although the number of furnaces now running is small, it is slowly increasing. The "Electro-Flex Steel Company" is putting down an initial installation of two one-ton Stassano arc furnaces at Dunston-on-Tyne for the production of steel by exclusively electrical methods.

A New Process of Electroplating

Plant has been erected in London for the demonstration of improved methods of electroplating, adopted by Mr. Q. Marino, which are apparently applicable to the deposition of all the ordinary metals and alloys on almost any metal and on non-metallic substances. It is noteworthy that aluminium

receives perfectly adherent coatings either of pure metals or alloys after a special preparatory treatment of brief duration—while for protecting iron with zinc the deposit obtained by the new process has obvious points of superiority over the coating obtained by the ordinary galvanizing. Tin-plate of very superior quality is said to be produced more economically than by the method in general use.

After certain chemical treatment which makes the surface conducting, without employing varnish and plumbago or other similar means, china and glass are coated with a film of adherent metal which can be polished, burnished and subjected to heat treatment.

Market Prices.

September, 1911.

Copper has remained fairly steady on the whole. On August 30th it was £56, and remained at this price till the 8th, then fell off, touching £54.7.0 by the 20th, but since recovered to £55 at which it closed.

Tin has been as erratic as usual. Starting at £192.10.0, it fell rapidly to £186.0.0 by the 5th, to £179.0.0 by the 14th, £176 on the 19th, and £171 on the 25th, closing, however, at £179.10.0, a recovery of nearly £9 in 5 days.

Lead fairly steady, and strong, opened at £14.12.6, rose to £15 by the 14th, and continued rising to the close, at £15.10.0.

Hæmatite has been flat, with a rise towards the end of the month, opened at 61/-, rose to 62/3 and remained at this price till the 21st, closing at 61/9.

Scotch Pig has been rather lower, opening at 53/-, dropping to 52/9 by the 14th, and closing at 52/6.

Cleveland has also been depressed, opening at 46/10 by the 14th, it ultimately closed at 46/6.

India Rubber (Para fine) has fallen off, opening at 4/9½, it was 4/½ on the 14th and closes 4/8.

	£	s.	d.
Alum, lumps, loose, per ton.....	5.12.	6	
Antimony, black sulphide powder, per ton.....	22.	0	0
Borax, British Refined Crystal, per ton.....	16.	0	0
Copper Ore, 10 to 25 per cent per unit 9/1½ to....	9.	7½	
Copper Sulphate, per ton.....	20.	0	0
Carbolic Acid, liquid, 97/99 per cent, per gal.....	1.	7	
Creosote, ordinary good liquid, per gal.....	2½		
Camphor, 1-oz. tablets	1.	9½	
Caustic Soda Ash, 48 per cent ordinary, per ton....	5.10.	0	
Hydrochloric Acid, per cwt.....	5.	0	
Mica, small slab, original cases, per lb.....	6d.	to 2.	0
Naptha, solvent, 90 per cent, 160°C., per gal.....	0.	8	
Petroleum, Russian Spot	0.	5½	
Quicksilver, per bottle	8.12.	6	
Sal Ammoniac, lump, firsts, delivered U.K., per ton..	42.	0	0
Sulphate of Ammonia, f. o. b. Liverpool, per ton..	13.17.	6	
Sulphur-recovered, per ton.....	5.	0	0
Shellac, standard T. N. Orange Spots, per cwt.....	4.	4	0
Tin Ore, 70 per cent, per ton.....	£112 to	114.	0
Zinc, Vieille Montague	32.10.	0	
Platinum	9.	5	0

The differences over the month noted are:

	Rise	Amount.
Copper Sulphate, ton	0.15.	0
Carbolic Acid, gal.	1	
Platinum	12.	6
	Fall	
Copper Ore, unit	4½	
Sulphate of Ammonia, ton.....	7.	6
Tin Ore, ton	8.	0

The Braden Copper Company, operating in Chile, has recently completed the first unit of its new concentrating plant. The complete mill will have a capacity of 3000 tons per day, and the first unit of 1000 tons capacity is probably now in operation. The ore is low grade, averaging 2.7 per cent copper.

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Economics of Tube Milling.—The tube mill was first introduced into the metallurgical industry by Dr. Diehl in Kalgoorlie in 1892, but it was not until 1905 that the machine assumed any importance in metallurgy. About the latter year it was extensively introduced on the Rand, where it has continued to grow in favor. The latest reports from the Rand show that 218 tube mills were in use there in June 1911, compared with 178 in the same month in 1910. In spite of the wide use of these machines the most efficient manner of their operation has not been a matter of settled conviction, and consequently the tests made by H. STANDISH BALL at McGill University, Montreal, Canada, are very interesting and valuable. The full results of Mr. Ball's work have been printed in *Bulletin* No. 83, Inst. of Min. and Met. The paper is replete with tabulations and curves which cannot be reproduced here, but the following gives a concise summary of the main points established.

Four series of investigations were carried out to determine the most efficient feed, moisture, pebble load and speed. In each series one of these factors was varied while the other three remained constant. Stadler's proposal for estimating crushing efficiency, based on Kick's law, was adopted in the calculations, and standard screen of the I. M. M. were used in the grading analyses. A general summary of all tests is given in the accompanying table.

General Summary of Tests.

General Summary of Tests.									
								Net Output — 120 Work Done Grade per (Tons Unit. 24 Hrs.)	Rel. Mech. Eff. per H.P.
Test.	Pebble Load.	Pebble Vol.	Rev. per Min.	Speed Min.	Mois- ture, %	Feed Tons per 24 Hrs.	I.P.		
Feed tests.									
P	1,200	0.5	41	369	38.0	7.2	6.3	3.89	2.27
C	1,200	0.5	41	369	38.0	9.6	5.0	3.36	2.34
B	1,200	0.5	41	369	38.0	12.6	5.1	2.82	2.95
O	1,200	0.5	41	369	38.0	14.4	5.6	2.91	3.31
*A	1,200	0.5	41	369	38.0	18.6	5.8	2.58	3.96
R	1,200	0.5	41	369	38.0	23.0	6.6	2.09	3.79
Moisture tests.									
D	1,200	0.5	41	369	30.0	12.6	5.7	2.99	2.93
*B	1,200	0.5	41	369	37.7	12.6	5.0	2.82	2.95
E	1,200	0.5	41	369	50.0	12.6	6.2	3.05	2.95
F	1,200	0.5	41	369	58.0	12.6	6.0	2.84	2.97
Pebble load tests.									
*G	900	0.37	41	369	38.0	12.6	4.9	2.75	2.65
B	1,200	0.50	41	369	38.0	12.6	5.1	2.82	2.78
H	1,500	0.63	41	369	38.0	12.6	6.9	3.31	3.40
Speed tests.									
T	1,200	0.5	33	297	38.0	7.2	6.8	4.19	2.45
*Q	1,200	0.5	37	333	38.0	7.2	5.1	4.17	2.59
P	1,200	0.5	41	369	38.0	7.2	6.3	3.89	2.27
S	1,200	0.5	46	414	38.0	7.2	7.5	4.03	2.48

*Most efficient test of series.

Series I.—Tests to determine the most efficient rate of feed. From the results obtained it was found that a feed of 18 tons per 24 hours is the most efficient one for the laboratory mill, any larger feed apparently causing the crushing efficiency as well as the relative mechanical efficiency to decrease. (The tube used in the experiments had outside dimensions as follows: 4 ft. 8 in. by 3 ft. 5 in. The lining was sillex brick set in cement.)

This is a most interesting point, as a critical feed is indicated, above which it seems probable that the fine grinding properties of the mill decrease in proportion approximately to the increase of feed. The theory of a critical feed may have an important bearing in modern practice, as it would seem that more efficient working, with a greater output of slimes would be obtained if a smaller feed were used. In South Africa the tendency at present is to increase the feed in the 5½ by 22 ft. mills, and although they are now feeding at the rate of 400 tons per 24 hours, the belief is held that even heavier feeds would result in increased grinding. The results of the present investigation show the danger of carrying this too far. Comparing the South African mills with the laboratory machine, it appears that the former would give better results if fed with about 300 tons per day instead of 400.

Series II.—Tests to determine the most efficient percentage of moisture in the feed.

The outstanding feature of these tests is the great efficiency of 37.7 per cent moisture over all others. This fact coincides with tests carried on in other parts of the world. In those conducted by Mr. Walter Neal the ideal dilution was determined to be in the neighborhood of 39 per cent moisture, although he believes that the critical point will vary with the size and nature of the mill. The results of the author's tests tend to disprove this belief. Sherrod found that the mill efficiency rose when the moisture was present to the extent of 40 to 45 per cent. G. O. Smart found the best moisture to be 38.5 per cent, while H. W. Fox discovered that, taking the power consumption in conjunction with the fine grinding, the most efficient moisture would be 39.6 per cent. Prof. J. W. Bell suggests that the critical moisture point depends on the percentage of voids in the sand, and that in all probability when the percentage of moisture is numerically equal to the percentage of voids in the sands inside the mill, the moisture is critical.

In connection with the tests on moisture, some novel work was done. From the first series of tests, on the variation in feed, it had been determined that the mill assumed uniform conditions after a period of about 35 minutes from the start of the test. This fact gave rise to the following idea, viz., if the mill was run for a given time at a certain fixed moisture until the conditions had become uniform and the moisture was then suddenly raised to another fixed point without stopping the mill, but allowing it to run for the same period, would not one be justified in assuming that the samples and power readings taken during the transition period of the mill from one state of normal conditions to the next, are representative of its product caused by the amount of moisture in it at that exact moment at which the sample was taken.

To test this theory, samples and power readings were taken at intervals of five minutes throughout three tests and were most satisfactory. It was found that whereas the mill took 30 minutes to adjust itself after the moisture had been changed from 30 to 50 per cent, it took only 12 minutes to assume uniform conditions from a change of 50 to 58 per cent, thus indicating that the time of change was proportional to the amount of change of moisture. With this fact so clearly demonstrated it was a simple matter to calculate what percentage of moisture the mill contained at the precise moment when a transition sample was taken, e. g., since the first transition period was 30 minutes and five samples were taken, the corresponding percentages of moisture would be $33\frac{1}{3}$ per cent, $36\frac{2}{3}$ per cent, 40 per cent, $43\frac{1}{3}$ per cent, $46\frac{2}{3}$ per cent. In the curves plotted by the author, the points obtained for these transition samples coincided with remarkable regularity with the curve drawn through the main points obtained for the actual tests. Hence the adoption of this method was more than justified, as it saved considerable time, material and labor, and gave a number of moisture efficiencies which otherwise would have been impossible to obtain.

Series III.—Tests to determine the most efficient pebble load. The result of this test indicated that the most efficient pebble load is about $\frac{7}{16}$ of the volume of the mill, whereas, disregarding power, the most effectual crushing load is about 0.6 of the volume which corresponds to present Rand practice. Whether it would pay to sacrifice power and gain fine grinding, or vice versa, would of course depend upon the conditions of the plant.

Series IV.—Tests to determine the most efficient speed of mill. These tests were most interesting as they proved rather conclusively that 37 r. p. m., at a peripheral speed of 333 ft. per minute, was the most efficient, both in relative mechanical efficiency and grinding efficiency; thus demonstrating that at that speed the grinding was decidedly better and the power required less. If this speed is exceeded or decreased the crushing effect of the pebbles is much reduced, while the power required is

increased. The efficient speed determined by the tests, viz., 37 r. p. m., compares with other determinations as follows: Davidson, 34; White, 39; Richards, 38.

General Conclusions.—In connection with the curves obtained from the different tests it is the author's belief that there is a possibility of deducing from them the probable results that would be derived by running the mill under different conditions. It seems that if one complete series of tests was run with a constant feed, the other factors being varied and the respective curves obtained, one would be able to deduce from them all the necessary information regarding any feed, provided that one test at that particular feed had been carried out.

Factors in Mexico's Progress.—Twenty years ago the bulk of Mexican silver ore was treated by the Patio process. Attempts were made frequently to introduce the American pan system, but for various reasons it was not successful. The non-adaptability of the ore; its irregular supply and the lack of mechanical knowledge stood in the way of progress. Changing conditions, however, forced new methods into Mexican metallurgy and have made it what it is to-day. In the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, VICTOR M. BRASCHI discusses the influence of some of the factors in Mexican progress. He regards the smelter, electricity and the cyanide process as the three most important of these.

The smelter first proved a benefactor in eliminating the long time usually required for recovery of the value from ore. Instead of the thirty days required for treating the torta, the smelter bought and paid for the ore on delivery. This resulted in abandoning to a large degree the Patio process, and most of the ores were diverted to the smelter. To quote Mr. Braschi's words, "The smelter began by helping the Mexican industry, but it was ending by throttling its very life." It had not only established a monopoly in ore treatment, but it had bought many large mines, and was making the lion's share of the profit.

About this time the cyanide industry was firmly established in the old mining district of Guanajuato, and the development of hydroelectric power in Michoacan combined with the cyanide process to point the way to freedom of the miner from the smelting process. Both of these improvements made much ore available that formerly could not stand the freight and treatment charges of the smelter. Not only ore in place, but much that was in the stopes and on the dump was treated at a profit.

It is Mr. Braschi's opinion that a modern school of mines and metallurgy will be the next step in the development of Mexico's wonderful mineral resources, and that chemical engineering also will be a great aid in utilizing many nonmetallic resources.

Continuous Cyaniding at Natividad, Mexico.—In a paper read at the Wilkes-Barre meeting of the American Institute of Mining Engineers, in June, 1911, Mr. HUNTINGTON ADAMS presented the results of continuous agitation in Pachuca tanks at Natividad, Mexico.

The difficulties encountered in intermittent filling and agitation in tall tanks of the Pachuca type are principally those due to the accumulation of pulp in the cone of the tank. Therefore a continuous flow of pulp from tank to tank has been devised and put in operation in several places. In this way the time of agitation of all the pulp is longer than by intermittent filling, and all particles of ore have the best opportunity to come in contact with the circulating solution.

The main point to be observed in continuous flow is that the composition of the pulp as regards ore and solution shall be maintained as uniform as possible. Otherwise a classification will result and in some tanks heavier particles will accumulate while the lighter particles pass on through the system. The only sure method of obtaining a uniform flow through the system is by arranging the overflow connections from tank to tank so that they will sample the overflow of the air lift pipe. This sampling should make a clean cut of the whole thickness of the air lift discharge. Some previous installations have not ob-

served this point, their connections being placed in the sides of the tank.

At the Natividad mine the concentrate is cyanided with the rest of the pulp after being ground very fine. Johnston vanners concentrate the slime overflow from Dorr classifiers, the concentrate being returned to the tube mill and reground until too fine to be caught on the vanners.

The arrangement of tank connections is as follows: A drop of 4 in. is used from tank to tank, and the central air lift tubes are cut down or added in order to give this drop. Wooden boxes 7 in. wide, 10 in. long and 6 in. deep are fixed against the central tubes with their tops flush with those of the tubes, and 4 in. pipes placed horizontally pass from the bottom of each box to the next tank in series. Bypass pipes fitted with valves, join each pipe connection with the next in the series, and the 4 in. drop from tank to tank is made in the by passes. Since the level of the inflow pipe in each tank is 2 in. below the tops of the air-lift tube and of the overflow box connected with the outflow pipe, but is slightly above the pulp level in the tank, no part of the pulp entering can pass out of the tank without first gravitating to the bottom, and risen through the air-lift tube, thoroughly mixed with the whole content of the tank.

On the tops of the overflow boxes are sliding iron covers which open at right angles to the direction of the overflowing stream. The regulation of the flow from tank to tank is done entirely by means of these covers, and the valves are used only when it is desired to bypass the tanks. The boxes are sufficiently large for the tonnage passing through agitation, so that the covers need be opened only an inch or two. The openings which sample the pulp stream are thus rectangular, with their long axes parallel to the radial overflow at these points; and while theoretically this is not so correct a shape as would be a sector of a circular ring, in practice it has been found to offer no difficulties.

As the last tank in the series discharges to the pulp tank of a Moore filter, where an intermittent feed is important, it is arranged with two pairs of sliding doors on the central air-lift tube so as to permit agitation at various levels. The pulp is drawn off intermittently to the filter through the bottom discharge openings of the tank, either by hydrostatic pressure in the tank if full enough, or by centrifugal pumps. All the tanks retain their bottom discharge connections with the pumps which lift to the filter, so that whenever it is necessary to empty any of them they may be cut out of the series and discharged to the filter.

The advantages of the Natividad installation over those of Grothe and Kuryla seem to be: Greater simplicity of installation; greater accessibility for handling and supervision; no plugging of the pipe connections can occur and no compressed air connections are necessary to free them; no classification is likely to occur as the boxes by which the flow from tank to tank is regulated provide convenient means of accurately sampling the mixed pulp as it overflows the air-lift tube, and the proportion of solution to slime remains the same throughout the whole system.

Physical Chemistry.

The Specific Heat of Solids at Extreme Low Temperatures.—In 1905 Dewar published a series of interesting results on calorimetric measurements carried out at temperatures of solid carbon dioxide and liquid air. (See *Chemical News*, 92, 181). Recently HERMAN BARSCHALL reconstructed the Dewar apparatus so as to ensure greater accuracy in the determinations. It consists essentially of three Dewar vessels, one inside of the other. Details and cuts are given in *Zeitschrift für Elektrochemie* of May 1. Only a few grams of the substance, whose specific heat is to be measured, are required to obtain very accurate results. The author found the following values for the specific heat at 183°C: lead, 0.0294; silver, 0.0492; cadmium, 0.0503; sulphur, 0.119; mercury, 0.0316; bromine, 0.073; mercuric iodide, 0.0375; lead iodide, 0.0377; lead bromide, 0.0463.

The Heat of Evaporation of Oxygen.—H. Barschall has

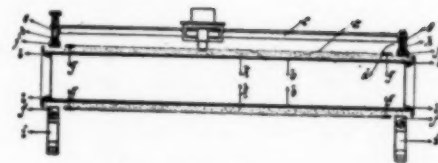
made a new determination of the heat of evaporation of oxygen (*Zeitschrift f. Elektrochemie*, May 1.) The average of six experiments is 51.30 calories per gram of oxygen boiling at ordinary atmospheric pressure. This value approaches that of Alt 50.97 rather closely, but is considerably smaller than the figure 61.0, published by Shearer in 1903.

Improved Laboratory Rheostat.—A German firm has recently put on the market a new porcelain tube resistance unit particularly adapted for laboratory purposes. As is shown in the accompanying figure the porcelain tube *k* is lined inside with a metal tube, the contact between metal and porcelain being the best possible. The resistance wire, wound on the outside of the porcelain tube, can be kept at temperatures approaching red heat without serious effects on the tube. The rheostats may be had in lengths of 10 to 100 centimeters. (*Zeitschrift für Elektrochemie*, June 1.)

Chemical Engineering.

Wislicenus Smokestack for Fume Dilution.—Investigation has pretty clearly established the maximum quantity of noxious gas which vegetation can successfully withstand. For SO_2 in relation to sensitive trees such as the spruce, this density is one part of gas in 500,000 parts of air. This is a small percentage, but is not much less than exists in some of our industrial towns other than smelting centers. In the latter the percentage rises much higher, and the necessity of diluting the gases in some manner becomes more important. To this end DR. WISLICENUS, of Germany, has invented a smokestack designed to aid in rapid dilution of discharged gases. The full description appears in the *Zeitschrift Ver. Deutscher Ing.*, June 10.

If stack gases are to be diluted so that they will not be injurious they must mix with large volumes of air at the time of their discharge from the stack, as it is well known that from



RHEOSTAT.

high stacks especially the smoke will persist in density for long distances. It is Dr. Wislicenus' idea that if the upper part of the stack were built in lattice form, with a large number of apertures for escape of the gases, a more rapid dilution would occur. Eddies would be formed which are most favorable to dilution. Consequently he has experimented with models and on a large scale, and has designed several forms of lattice structure which break up the dense smoke and mix it immediately with larger volumes of air than would be the case if the dense smoke issued in the ordinary way. One form of lattice consists simply of radial openings, another of tangential openings, and still another with projecting and enlarged vents for steel stacks. Correct construction is necessary to obtain the best effect.

Removing Oil from Return Feed Water.—The presence of any considerable quantity of oil in return feed water is injurious to boiler tubes and plates, reduces the evaporative efficiency and aids in the deposition of scale. Various schemes have been proposed to overcome this, such as submitting the water to an electrical discharge between iron electrodes for the purpose of

coagulating the oil. This scheme, however, has been found to cause deterioration by corrosion, probably due to electrolysis.

A successful plan is described by C. E. CROCKER in the July 1911 *Journal of the Chamber of Mines of Western Australia*. It consists in treating the oily water with alum and common soda in the proportions of 0.35 lb. of alum and 0.30 lb. of soda per thousand gallons of water, for water containing 3.5 grains of oil per gallon. The quantities of chemicals used are, of course, proportioned to the quantity of oil contained in the water. The apparatus should have sufficient capacity to allow the chemicals ample time to act. In the plant referred to the capacity is 50,000 gallons per 24 hours, and the tanks and filters are arranged so that five hours are required for the water to pass through.

The alum and soda are mixed in tanks and allowed to drip into the incoming oily water. The precipitated water is filtered through coke, charcoal, shavings, or any open porous bed. The filters are cleaned as often as necessary. The cost is about 2.3 cents per 1,000 gallons. The plant has been in operation for two years, and the boilers require less cleaning than formerly.

Ozone.

Ozone Generators.—Prof. C. HARRIES, of Kiel University, reports in the *Zeitschrift für Elektrochemie* of August 1, results of experiments made with a Siemens and Halske ozone generator. When using pure oxygen a maximum output of 18 per cent to 19 per cent ozone was obtained at a potential of 7400 volts alternating current, and frequency of 100 at 20° C. The ten glass tubes of the generator were mounted in a wooden instead of the usual metallic frame, and as a consequence an appreciable increase in efficiency was obtained. For parallel connected tubes the most favorable gas velocity was 60 liters per hour and for series connected, 8.6 liters per hour.

The efficiency of the generator increases after repeated using. When air or pure nitrogen was passed through the tubes an intensive blue luminescence appeared which was absent, however, when pure oxygen was used. This blue luminescence is not to be confused with the blue color of highly concentrated ozone gas; it is not due to the presence of traces of oxides of nitrogen, as Warburg and Leithaeuser had assumed, but is the inherent color of the gas. Ozone will react fairly readily with unsaturated hydrocarbons resulting in addition products, such as ethylen ozonide and propylenozonide, that are highly explosive. The study of these compounds lead to the assumption that ozone gas contains quadri-atomic oxygen besides the tri-atomic. By careful fractional distillation and analysis the author succeeded in verifying his assumption. The formula of oxozone is either O_4 or O_6 .

GIDEON LECHNER has investigated the efficiency of a Berthelot ozonizer. With direct current he finds that the ozone output is directly proportional to the frequency of the interrupter. Increasing the closed-circuit interval of the interrupter increases the mean current intensity and at the same time almost proportionately the ozone output. An alternating current of 42 cycles is more efficient than a direct current of frequency 42 owing to the flatter form of the wave and consequent greater mean current intensity. (*Zeitschrift für Elektrochemie*, June 1.)

Minimum Potential of Silent Discharge.—Prof. E. H. RIESENFELD, of Stockholm, investigated the behavior of a large number of gases when passed through a generator tube of the Jannasch type. The results are recorded in the *Zeitschrift für Elektrochemie*, September 1. The author found that the relative minimum potential necessary to bring about the silent discharge is well defined for every gas and independent of the size or shape of the generator tube, of the velocity of the gas and of the frequency of the current. The relative potential is defined as the potential expressed in terms of the oxygen potential taken as unity. The minimum potential of a gas is proportional to its continuous potential or potential required to effect a steady continuous discharge. The author's values for the relative minimum potential are: Hydrogen, 0.77; nitrous oxide, 0.85; ammonia, 0.97; oxygen, 1.00; carbon dioxide, 1.03; carbon

monoxide, 1.032; acetylene, 1.11; ethylene, 1.11; nitrogen, 1.16; hydrogen sulphide, 1.27; hydrochloric acid gas, 1.27; chlorine, 2.42. The luminescence color of each gas is very distinct and different from that of any other. Hydrogen shows a pale blue, oxygen a deep greenish blue, nitrogen a decided purple, and chlorine a very intense green luminescence. There does not seem to be any direct relation between the phenomena of spark discharge and those of the silent discharge.

Electrolytic Production of Ozone.—The electrolytic production of ozone was recently investigated by E. H. ARCHIBALD and H. VON WARTENBERG (see *Zeitschrift für Elektrochemie*, September 15.) Dilute solutions of sulphuric acid were electrolyzed with direct current and with alternating current superimposed. Owing to the depolarizing effect of the alternating current the output of ozone was greatly increased over that obtained with direct current alone. In some cases the increase was three hundred fold. Platinum electrodes were used. By keeping the intensity of the direct current constant and increasing the alternating current step by step the amount of ozone rose steadily. The maximum output was obtained at about 6 amperes, alternating current, for all values of direct current between 0.25 and 1.00 amperes. An increase in the frequency of the current was accompanied by an increase in the ozone output. The principle involved is commercially utilized by the Norddeutsche Affinerie in their gold-silver parting process (Wohlwill process.) Formerly, when using merely direct current, a considerable part of the chlorine escaped the bath as gas instead of combining with the silver; this loss, however, is now entirely avoided by superimposing an alternating current. (See our vol. viii, p. 82.)

Recent Metallurgical Patents.

Iron and Steel.

Nodulizing and Desulphurizing Kilns.—For the purpose of desulphurizing and nodulizing iron ores in one rotary furnace, Mr. DAVID BAKER, of Philadelphia, Pa., has patented a furnace. The invention consists primarily in constructing the furnace in two sections, in one of which desulphurizing takes place, and in the other nodulizing or sintering. Invention also lies in combining with a rotary furnace a scraper or cleaning device to prevent the accumulation of rings in the hottest part.

The desulphurizing end of the kiln consists of a cylindrical furnace, having a slight pitch so that the ore will travel through the furnace as it revolves. The nodulizing portion consists of an extension of the kiln in the form of a truncated cone, which is relatively shorter than the cylindrical portion. The smaller end of the truncated cone encircles the end of the cylindrical portion so as to receive the ore discharged therefrom. Different speeds are imparted to the two parts of the furnace, so that the desulphurizing end runs slower than the nodulizing portion, there being less heat and longer time required for desulphurizing and higher heat and shorter time for nodulizing. A satisfactory speed is 1 r.p.m. for the cylindrical portion and 4 r.p.m. for the truncated cone. The scraper is a bar sustained in such a position that it is in close proximity to the side of the nodulizing kiln. It is cooled. (1,004,340, Sept. 26, 1911.)

Still another means for preventing the formation of rings in sintering furnaces has been patented by Mr. HARTLEY C. WOLLE, of Westmont Borough, Pa. It consists in embedding cooling pipes in the interior walls of the kiln. The object of the cooling pipes is to so cool the furnace in their vicinity that a continuous ring can never form. Sintered accumulations may form, but they will be in the form of an arch without a keystone, and in the natural revolutions of the furnace they will tend to fall off. One or more of these cooling pipes may be embedded in the furnace wall, thus causing one or more breaks in the continuity of any ring which might form. The coolers are cooled by circulating air or water through them. (1,005,396, Oct. 10, 1911.)

A similar effect is obtained by LEWIS P. ROSS, of Standish, N. Y., by supporting a T-bar in the furnace to detach material adhering in the form of a ring or annulus. The bar is suspended so that it is not rigid, and is cooled by water pipes running its length. (1,004,085, Sept. 26, 1911.)

Electric Steel Furnace.—Among the more recent combination furnaces for electric steel refining is that of Hans Nathusius, which has been employed in commercial tests at Friedrichshütte, Germany, (our vol. viii, p. 490 and 546). It combines heating from the top by means of arcs from vertical electrodes with heating from the bottom by means of

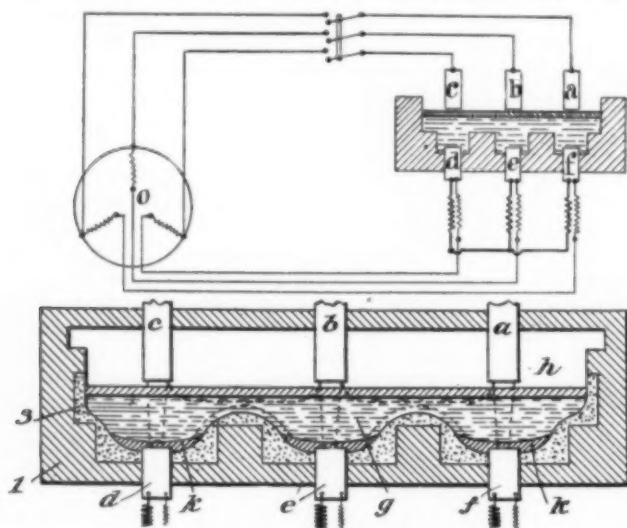


FIG. 1. ELECTRIC STEEL FURNACE.

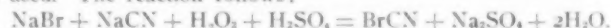
metallic electrodes embedded in the bottom and separated from the furnace charge by a so-called conductor of the second class, as in the Roehling-Rodenhauser furnace. The object is to heat from the top when the refining depends on a very high temperature of the slag and to heat from the bottom at other periods when this may seem desirable. To increase the heating effect from the bottom, the arrangement shown in Fig. 1 is employed. With three-phase currents, as shown in the upper diagram, this is accomplished "by dissolving the nodal point of the machine and connecting the three inner ends which otherwise form the neutral point, to the bottom electrodes, while the outer ends are connected in the usual manner to the three surface electrodes. By means of this connection the point is transferred into the metal bath itself."

"This heating action in the metal bath, or the increase of heating action due to resistance heating in the metal bath or in the hearth role, by increasing the current, can also be advantageously obtained by connecting in front of the bottom electrodes a transformer which transforms the current from the main furnace source of current to a very high strength of current with a correspondingly small pressure, so that in addition to the primary current a current of any desired strength is supplied by the bottom electrodes direct to the metal bath or to the hearth sole." The lower diagram shows a lining *k* formed of a conductor of the second class disposed around and over the inner extremities of the bottom electrodes *d e f*. The upper electrodes are *a b c*, the hearth is *g* and the slag layer *h*. The refractory lining *3*, which is rammed between the bottom electrodes and which, when the furnace is in operation, is traversed by the current, consists of ordinary burned dolomite, being mixed with tar as binding agent. This is the so-called "black mass" which serves in a similar manner for the preparation of basic open-hearths.

The transformer is best built directly under the electric furnace as a part of it to permit easy bottom connection. (1,000,838, August 15, 1911.)

Gold and Silver.

Improved Method of Forming a Halogen Cyanide.—In cyanidation it has been found advisable in some cases to make use of bromocyanide, or similar halogen compound. The usual method of forming this compound has been by mixing sulphuric acid, potassium cyanide and an alkali bromide and bromate. The mixture usually is made only when it is needed, and the procedure is one requiring care. It is the aim of Dr. HANS FOERSTERLING, of the Roessler & Hasslacher Chemical Company, of Perth Amboy, N. J., to provide a better means of forming halogen cyanides by melting an alkali halogen with an alkali cyanide in the proportion of one molecule each and permitting them to solidify. No decomposition takes place in this process and the double salt is used instead of making the mixture above referred to. In order to liberate bromocyanide, for example, from a mixture of sodium bromide and sodium cyanide, an oxidizing agent such as an acid solution of hydrogen peroxide is used. The reaction follows:



By this method it is possible to provide cyanide mills with a correctly proportioned mixture ready for use when dissolved and mixed with an oxidizer. (999,215, Aug. 1, 1911.)

Precipitation of Gold and Silver Prior to Filtration.—The use of aluminium as a precipitant for gold and silver from cyanide solutions has been proposed before, the aluminium being merely a substitute for zinc. Mr. JAMES E. PORTER, of Syracuse, N. Y., proposes to use aluminium or its suitable alloys to precipitate gold and silver from cyanide solutions without filtering the solution from the ore pulp.

The ore is finely ground and agitated for a short time with caustic soda to neutralize cyanides and to give the ore a protective alkalinity. Cyanide is then added and the mixture agitated, preferably with aeration and by the use of so-called silica sponge, until all the gold and silver are in solution. If the agitation and aeration are well done the inventor states that sulphides can be effectively treated in the raw state.

Strips of aluminium are then immersed in the pulp while the agitation is continued; the immersion being accomplished in any suitable manner. The dissolved metals will precipitate on the aluminium, which is periodically removed and cleaned by wiping or scraping. As long as any precipitate forms the operation is repeated. The precipitation of gold liberates potassium cyanide and cyanogen; some of the aluminium goes into solution as potassium aluminate with liberation of hydrogen, and the cyanogen and hydrogen combine with the alkali present to form potassium cyanide, thus regenerating the cyanide combined with the precious metals. (1,002,446, Sept. 5, 1911.)

Direct Cyanidation of Sulphides.—In cyaniding sulphide ores containing silver, or gold and silver, with the latter usually predominating, silver sulphide goes into solution in potassium cyanide, forming potassium silver cyanide and potassium sulphide. The solvent action is, however, limited to the state of equilibrium which soon occurs, and after that point has been reached the reaction is reversible and silver again goes into solution. This phase will occur before all the silver has been dissolved. It is the object of an invention of JAMES E. PORTER, of Syracuse, N. Y., to provide means whereby the sulphide of potassium formed may be oxidized to thiosulphate, thereby avoiding reprecipitation and aiding in the extraction.

Mr. Porter shows that some form of oxidation is necessary to accomplish this, and proposes to obtain it by energetic and prolonged agitation with air. The conditions of success which he imposes are (1) fine grinding of the ore; (2) suspension of the ore in cyanide solution; (3) thorough permeation of the mass by minute bubbles of air which will remain in the liquid for some time and not rise quickly to its surface, and (4) operation at a temperature of about 180 deg. Fahr. to 190 deg. Fahr. Under these conditions he claims that it will be impossible for soluble sulphides to remain as such, their oxidation following

closely upon their formation, and equilibrium between dissolved silver and dissolved sulphide cannot exist. In this process the inventor prefers to use silica sponge as the medium for atomizing the air. (1,002,447, Sept. 5, 1911.)

Impact Filter.—A filter utilizing centrifugal force in the separation of cyanide solution and ore pulp has been patented by THOMAS H. NEIKIRK, of Midas, Nev. As shown in Fig. 2 the filter consists of a revoluble drum in the shape of a truncated cone, suitably mounted on a hollow shaft and driven from a pulley at 4. Within the drum is a filtering medium against which the incoming pulp is thrown from a number of radial members mounted on the hollow shaft. The filter is designed to work continuously as regards the formation and discharge of cake, and with the minimum of attendance.

The operation of the machine is as follows: Slime, barren solution or wash water are introduced into the filter through the pipes 18, 19 and 20, respectively, each being properly controlled by valves as shown. When the valve on the slime line is open slime enters from pipe 18 to the hollow shaft 3, and is distributed by centrifugal force through connecting radial arms

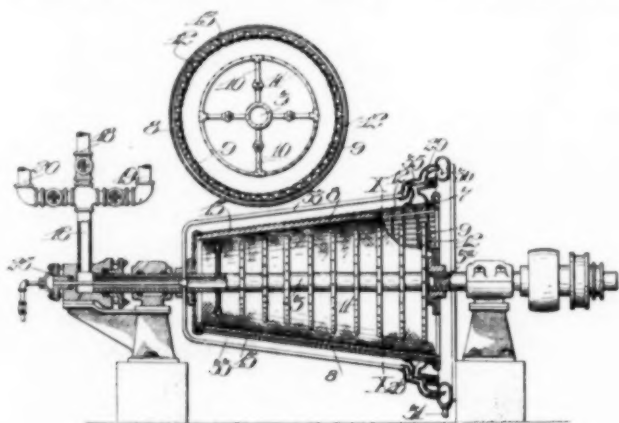


FIG. 2. IMPACT FILTER.

to perforated rings 11. The pulp issues from the ring perforations and is forced against the surrounding filtering medium 9, which is supported on a grooved surface 8. The liquid is thus separated from the solids and passes by perforations 12 into an annular channel 13, from which it is eventually discharged into an annular collector 28, which is fastened to the outside of the shell at its larger end. From this collector the solution is finally directed into a drain pipe 31, from which it is conducted to the zinc boxes. The solids accumulate on the filter, and when a cake of sufficient thickness is formed, the valve controlling the incoming slime is closed and barren solution and wash water introduced in a similar manner. The cake having been washed, it is now dislodged from the filter surface by means of a fluid introduced under pressure through the pipes 26 and 33, which connect with the annular channel beneath the filter. Automatic valves 35 prevent the pressure water from finding its way into the collectors which were used for discharging the filtered gold-cyanide solution. The pressure water and filtered pulp thus mix in the cone and are discharged at the lower end of the drum, after which the cycle is repeated.

The impact of the slime pulp against the filtering medium is designed to increase the efficiency and capacity per unit of filtering area. Another important feature is the relative size of the machine to the common filtering apparatus now used. (1,000,540, Aug. 15, 1911.)

Zinc, Lead and Copper

Zinc Furnace.—In the accompanying illustration, Fig. 3, is shown a cross-sectional view of a furnace for the distillation of zinc ores, designed and patented by FRANK LAURENT CLERC, of Boulder, Colo. The novelty of the invention lies in the means used for applying heat to the charge of zinc oxide

and reducing substance, and in the proposed manner of introducing the charge. The furnace consists of a retort *a* and heating chamber *b*. The retort has a port *d* for the exit of zinc vapors, and a feeding spout *e* for supplying the charge to be smelted. The bottom of the retort consists of a body of molten metal *f*, such as copper or an alloy of copper and zinc, or a

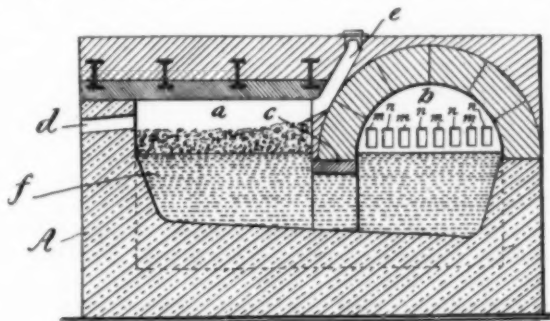


FIG. 3. ZINC FURNACE.

ternary compound which has a melting point higher than the temperature at which zinc is reduced from its oxide by carbon, and which has a high heat conductivity. This body of metal is kept in a molten condition by hot gases from a suitable source of combustion, and the heat is thus conducted through the molten metal to the charge of ore and fuel as shown on the hearth. The reduced zinc vapors pass out through the port *d* and are condensed in a suitable manner. The accompanying carbon monoxide can be collected and burned for any useful purpose. (1,002,037, Aug. 29, 1911.)

Precipitating Copper from Solutions.—The usual manner of recovering copper from dilute solutions such as mine water or water flowing over old copper dumps, is to precipitate the red metal on scrap iron. As is well known, the quantity of iron required is several times that indicated by theory, and the precipitate is not high grade. Mr. E. E. SLAUGHTER, of Clifton, Ariz., has patented an improvement in this method, having assigned the same to W. L. Austin, of Riverside, Calif. Mr. Slaughter prefers to precipitate copper by using a mixture of iron and carbon in the form of small pieces of coke. He thus gains the advantage of an electrolytic action, and states that the precipitated copper is free from basic iron salts and of higher grade than that obtained by precipitation on iron alone. (1,001,466, Aug. 22, 1911.)

Refining and Desilverizing Lead.—The crystallization process used in the refining of lead bullion carrying gold and silver are based on the fact that when such bullion is melted and cooled down to nearly the solidifying point and stirred by hand or otherwise crystals of lead form which are freer from silver and other impurities than the liquid which permeates the mass of crystals. It is customary in this process to draw off the fluid from the crystallized lead, and repeat the process. Usually twelve treatments are required to give commercial lead as one product and a high-grade bullion suitable for cupeling as the other.

Mr. GEORGE P. HULST, of Omaha, Neb., has patented means for handling the semicrystallized mass in a press, by means of which the liquid is squeezed from the crystals, rendering the latter drier and concentrating the impurities in the liquid. By this procedure the number of crystallizations is reduced to six, as compared with twelve as usual. (1,001,525, Aug. 22.)

Cyanidation of Silver Ores at Cobalt.—At the Nova Scotia mill in the Cobalt, Canada, district cyanidation is in use, as noted in the *Mining Magazine* for September, 1911. The ore is crushed in stamps in cyanide solution, and concentrated without classification on Deister No. 2 tables to remove the metallics and rich sulphides. The table tailings are classified, reground in tube mills, thickened in Dorr thickeners, agitated in Trent agitators, and filtered by the Moore process. Precipitation of silver is by the Merrill zinc dust method.

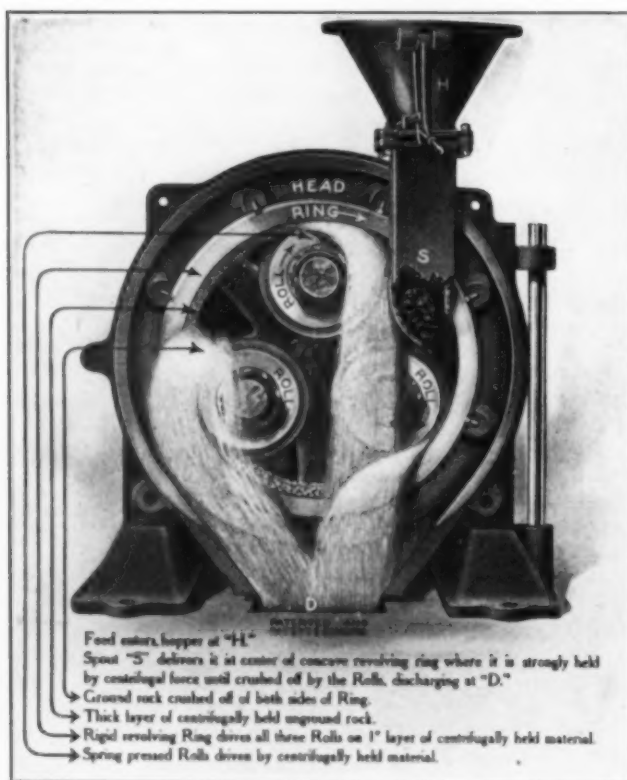
Combination of Ring Roll Mills and Tube Mills in the Grinding of Cement Clinker.

One of the most important problems in many branches of metallurgical and chemical engineering is the proper combination of different grinding machines so as to grind a given material at the lowest cost. Under different conditions the best combination will differ, but in many cases it will be found advantageous to subdivide the whole grinding operations into a series of different ones, carried out in different machines, instead of trying to finish the whole job in one machine.

A case in point is the grinding of cement clinker, and in this respect some tests made by Sturtevant mills are of considerable interest.

The usual practice in reducing Portland cement clinker is to grind the clinker in one ring-roll mill to 20 mesh and finish in three tube mills.

The alternative method, which was compared with that just



RING ROLL MILL.

mentioned, is to grind the clinker in two ring-roll mills to 80 mesh and finish in one tube mill to 83 per cent—200 mesh.

In each case the ground product of the ring-roll mill was raised by means of an elevator to Newaygo screen separators to separate the fine material, which then underwent finishing treatment in the tube mill.

The comparison between the two methods extended to the establishment of comparative figures, firstly, for wear and tear of the machinery, and secondly for power consumption. Of course, the following figures are only comparative, using the same hard clinker.

As to the *cost of maintenance*, the wear and tear of the one ring-roll mill grinding 45 bbl. to 20 mesh per hour represents an expense of 0.076 cent per bbl. The wear and tear of the Newaygo separator, handling 45 bbl. per hour, was 0.022 cent per bbl. The wear and tear of the three tube mills was 0.266 cent per bbl. Hence the total wear and tear of the ring-roll mill, separator and the tube mills (not including elevator) was 0.364 cent per bbl.

On the other hand the wear and tear of the two ring-roll mills, grinding 45 bbl. to 80 mesh per hour, was 0.152 cent per bbl. The wear and tear of the two separator screens was 0.044 cent per bbl. The wear and tear of the one tube mill was 0.089 cent per bbl. Hence the total cost of wear and tear of the two ring-roll mills, separators and the one tube mill was 0.285 cent per bbl. This method, compared with the first one, exhibited a saving of 0.08 cent per bbl. in the maintenance cost of the grinding machinery.

As to the *power consumption*, the one ring-roll mill in the first case requires 0.777 hp per bbl., the Newaygo separator 0.022 hp per bbl. and the three tube mills 5.666 hp per bbl. Hence the total power consumption of ring-roll mill, separator and tube mills is 6.465 hp per bbl.

In the second case the two ring-roll mills require 1.555 hp per bbl., the Newaygo separator 0.044 hp per bbl. and the tube mill 1.888 hp per bbl. Hence the total power consumption of ring-roll mills, separator and tube mill was 3.487 hp per bbl. This represents a saving of 2.987 hp per bbl.

Both with respect to maintenance cost and power consumption the second method turned out to be considerably more economical.

In this comparison the elevator has not been considered since its wear and tear as well as power consumption was taken as the same in both cases.

It is clear from the above comparative figures that the decisive item in the results of the tests was the decidedly high economy of the ring-roll mills and for this reason a description of the Sturtevant ring-roll mill to which the above figures relate should be of interest.

The adjoining illustration shows the construction. A heavy steel anvil ring is secured in a head supported by a horizontal shaft. Against the inner face of this ring hammer rolls are elastically pressed with great force and revolved by frictional contact with material on the ring.

Substances to be ground are fed (up to 1¼ in.) to the inner face of the rotating ring which holds them thereon by centrifugal force to be crushed as drawn under the rolls by ring rotations and discharged on both sides thereof.

The face of the ring is concave and the roll faces convex.

The roll mountings are on the massive door that forms one side of the mill casing, and they are swung away from the ring with the door when it is opened. Thus the whole grinding outfit is made accessible by opening the door. The roll shafts are as large as those of the driving wheels of a locomotive, and crushing pressures are greater. One adjusting screw on the outside of the door regulates the roll forces and gives the rolls an absolutely equal pressure. No rock can withstand these pressures.

When at work, the concave of the revolving ring is always covered with a thick layer of material fed thereto. A naked track is never exposed to the roll faces. They cannot damage one another. Rock is crushed down upon itself (between anvil ring and hammer roll) producing a maximum of fines, with least wear.

As there is a constant feed, while the mill is at work, of coarse and partly reduced material, so there is a constant drop of material crushed off to both sides of the ring by the rolls. This escapes, as in all mills of this class, from the bottom of the case, and is taken to a Newaygo screen to remove the finished product *as soon as made*. The tailings, separated by this screen, are returned to the ring (with fresh feed) for further reduction.

Thus the mill is always breaking down coarse rock upon tailings, producing a maximum output and forcing the rocks to crush one another.

It should be emphasized again that in the Sturtevant ring-roll mill the ring drives the rolls. They run like those of an automobile's roller-bearing. No roll drives are used nor do the rolls support the ring.

The Absorption of Hydrochloric Acid and Other Gases.

BY PERCY C. KINGSBURY.

Practically all of the muriatic acid made in this country is obtained from the decomposition of common salt by means of sulphuric acid or bisulphate of soda.

The hydrochloric acid gas evolved during the reaction is

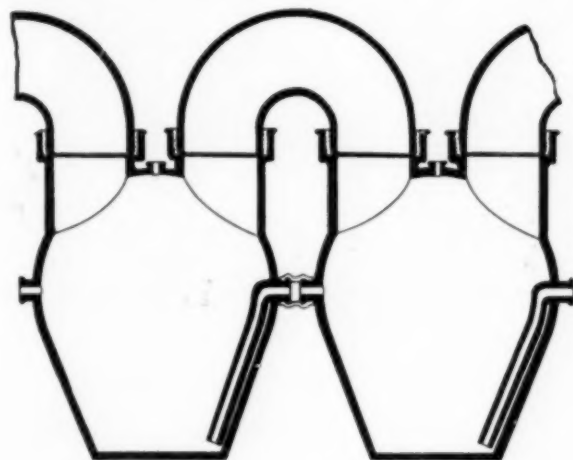


FIG. 1.—STANDARD FORM OF TOURILL.

first cooled and afterward dissolved by water. The cooling of the gas is effected by passing it through large pipes exposed to the air. The pipes are usually of fire clay and on account of the porosity of this material are boiled in tar before being installed.

The plant commonly employed for the absorption consists of a system of stoneware "tourills" connected in series by stoneware piping. A current of water or dilute acid passes through these vessels over the surface of which liquid the cooled gas circulates. One or two small wash towers usually complete the absorption equipment.

For many years little consideration was given to the design of an effective type of tourill for this purpose, chemical manu-

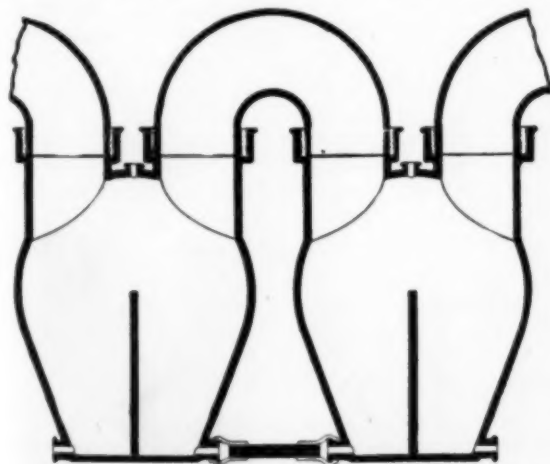


FIG. 2.—ANOTHER STANDARD FORM OF TOURILL.

facturers being content to use the standard form of vessel supplied by the stoneware manufacturers.

This form, illustrated in various modifications in Figs. 1, 2 and 3, has the advantage of being easy to manufacture, and, on account of its symmetrical shape, of resisting the expansion and contraction due to variations in the temperature to which it is subjected.

The principal function of the tourill is to dissipate the heat evolved during absorption as the amount of gas that can be

absorbed, or, in other words, the strength of the acid obtainable, decreases very materially with rise in temperature of the absorbing liquid.

An efficient type of tourill should, therefore, offer as large a cooling surface as possible compared with the volume of acid it contains. Those illustrated in Figs. 1, 2 and 3 are for all practical purposes spherical, and therefore from the above point of view of efficiency the least desirable, as of all geometrical forms the sphere has the least surface in relation to its volume.

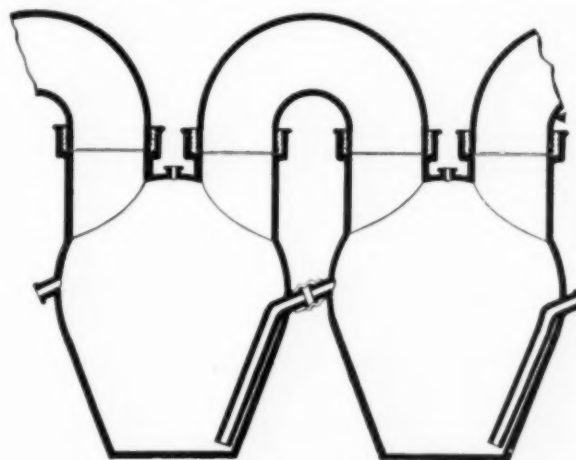


FIG. 3.—ANOTHER STANDARD FORM OF TOURILL.

This tourill has, moreover, several other disadvantages. The acid contained presents a comparatively small surface of contact with the gas to be absorbed, the vessel is difficult to clean out and cannot conveniently be water-cooled. The latter is an important consideration, as artificial cooling is necessary in working for acid of the highest possible strength, especially during the summer months.

These considerations led some acid manufacturers to replace the spherical tourills by vessels of more or less fanciful shapes, most of which proved unsatisfactory on account of the designers not taking into consideration the limitations of the material of which such articles have to be made.

The first practical improvement on the old-fashioned type is the Cellarius tourill illustrated in Fig. 4. Several of the advantages of this vessel are obvious. It can be completely immersed in water, offers a much larger cooling surface in comparison with the volume of acid it contains and also exposes a larger surface of the absorbing liquid to the action of the gas.

By connecting successive tourills in the manner shown in Fig. 5 the vessel can immediately be emptied and flushed out by simply reversing the glass "T" piece. It will be noted that the acid enters the tourill by one of the projecting tubulures and before leaving by the other must flow through twice the entire length of the vessel, passing in a shallow stream over



FIG. 4.—CELLARIUS TOURILL.

the arch at the back of the tourill. This insures an efficient cooling of the liquid and a progressive increase in the strength of the acid as it passes through the vessel.

The tourill withstands variations in temperature very well, as the form of a double arch enables it to take up considerable strain due to expansion and contraction of the material. This form is also advantageous from the point of view of mechanical strength.

One other practical advantage of this vessel is that the acid

connections are under the surface of the liquid, thus preventing the possibility of stoppage of the flow due to the formation of air locks, a matter of frequent annoyance in the old type of vessel.

The many advantages of the Cellarius tourill have received prompt recognition in Europe, where it is rapidly replacing the spherical vessel. Where such substitution has been made it has been found that one Cellarius tourill can replace two or three of the old-fashioned type.

No difficulty is experienced in connecting both kinds of ves-

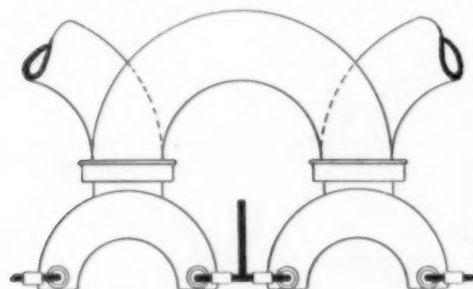


FIG. 5.—CONNECTION OF SUCCESSIVE TOURILLS.

sel to the same battery. The advantages of the new tourill can, therefore, be realized practically at little expense or disturbance of operation by substituting or adding a few Cellarius tourills at the beginning or end of a series.

The success of the Cellarius tourill for absorption purposes has led to the design of a modification of this vessel for the cooling and condensation of gases. This pattern is illustrated in Fig. 6 and it will be noted is merely an inversion of the type used for absorption, the advantages of which are retained. It has been employed for the pre-cooling of hydrochloric acid gas



FIG. 6.—MODIFICATION FOR COOLING AND CONDENSATION OF GASES.

before it enters the absorption battery and for the condensation of nitric acid and is a practical and efficient vessel for all purposes where acid gases are to be cooled or condensed to the liquid state.

The Cellarius tourill was exhibited at the Pittsburgh meeting of the American Institute of Chemical Engineers, December, 1908 (this journal, Vol. 7, page 89), by the Didier-March Company, New York, who are the manufacturers of these vessels in this country.

The Oxygen-Gasoline System of Lead Burning.

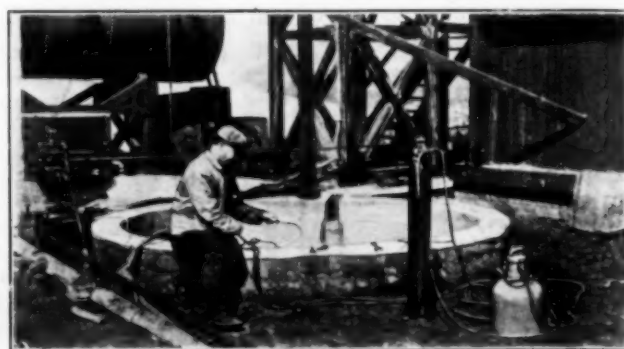
While in the old familiar system of "lead burning" hydrogen and air are employed, the new Howard system involves the use of oxygen gas saturated with gasoline. With this new system much more perfect combustion and higher temperatures can be obtained and it is claimed to combine the advantages of superior results with respect to quality of work, with greater economy, greater speed of operation and greater convenience in use.

The adjoining illustration shows the Howard apparatus in operation. It will be seen that there are only two principal parts, namely the gasoline saturator, weighing about 28 lb., and the oxygen cylinder weighing about 68 lb.

To the valve on the oxygen cylinder the oxygen regulator is attached. One outlet of the oxygen regulator is connected by a rubber hose to the blowpipe, while from the other outlet of the oxygen regulator another rubber hose leads to the inlet of the gasoline saturator. The exit gas from the gasoline saturator

passes through a hydraulic trap, which serves as a safety device in the case of a back-fire, and is then conducted through a rubber hose to the other nipple on the blowpipe. A proper burning mixture is produced at the blowpipe by manipulation of the gas and oxygen cocks.

There is nothing difficult about the operation. In fact, any workman conversant with the hydrogen-air system of lead



HOWARD LEAD BURNING APPARATUS IN OPERATION.

burning will experience no difficulty in operating the Howard lead burning system.

The advantages of the Howard system are essentially due to the simple apparatus required, as described. As compared with the old hydrogen-air system, the hydrogen generator is dispensed with. The air bellows or pump is dispensed with and consequently the services of a boy are not required. The apparatus is portable and may be carried anywhere, thus doing away with the very troublesome necessity of moving a heavy hydrogen generator and bellows from one job to another.

There is no apparatus to get out of order, involving expense, delays and repairs. No acid or corrosive substance is employed. As neither zinc nor sulphuric acid is used, there is no deleterious matter to be carried through the blowpipe to act injuriously on the lead seam.

The flame produced is absolutely clean and does not blacken nor tarnish lead, brass or copper. This is of special advantage in soldering the two latter metals, and also in repairing old lead work that is thin, as very little shaving is necessary.

A much better, hotter flame is produced than with hydrogen and air and much superior and faster work can be accomplished. The flame is so hot that even heavy lead in wet and cold positions can be burned "in situ" without preheating. A general idea of the comparative speed of lead burning by means of the Howard apparatus and the hydrogen and air method respectively may be gathered from the figures in the following table, in which the approximate foot run burnt per hour is given for various weights of lead sheet.

	Flat seam 20 lb. lead	Gasoline-Oxygen		Hydrogen-Air	
		20 ft. per hour	15	ft. per hour	
"	12 "	48 "	27	"	"
"	8 "	60 "	40	"	"
"	6 "	108 "	90	"	"
Upright "	20 "	5 "	4½	"	"

No gas is generated when the blowpipe is not in use, and consequently there is no waste of gas and no charge to withdraw overnight.

The apparatus can be recharged in about three minutes. This will appeal to lead burners who are familiar with the time taken to recharge the hydrogen apparatus.

The Howard system of lead burning is being introduced by the Linde Air Products Company, of Buffalo, N. Y., which firm has been so successful in the last years in developing new uses for oxygen and especially in popularizing the oxy-acetylene system of welding. It is hardly necessary to add that this company makes the oxygen from liquid air by the Linde process.

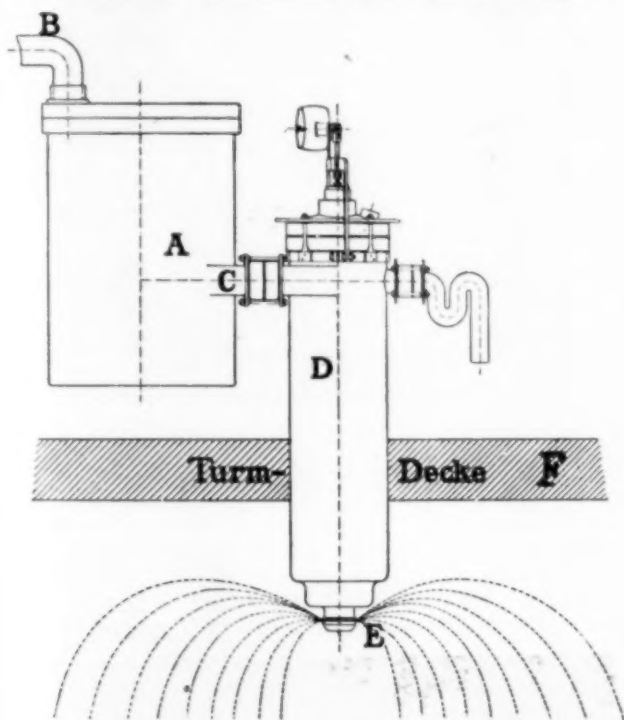
Novel Feeding Device for Glover and Gay Lussac Towers.

The adjoining illustration shows the "effector distributor" for tower liquid, *i. e.*, a new feeding device for Glover and Gay Lussac towers, built by the Deutsche Steinzeugwarenfabrik für Canalisation und Chemische Industrie in Friedrichsfeld.

Its chief advantages are that there are no valves nor other movable parts in it, that the liquid is distributed absolutely uniformly over the whole cross-section of the tower, and that only one opening in the top of the tower is required. Neither variations within rather wide limits of the liquid supplied to the tower per hour nor considerable impurities in the liquid are detrimental to the proper operation of the apparatus.

The "effector distributor" is especially suitable for the sulphuric acid industry, since on account of its simplicity and robustness of construction it permits a perfect and uniform distribution of the liquid over the whole tower cross-section in Gay Lussac and Glover towers under all conditions.

The vessel *D* is supplied with the liquid from the filling device



FEEDING DEVICE FOR GLOVER AND GAY LUSSAC TOWERS.

A which operates at periodical intervals. The liquid is gradually given off from *D* to the tower through the spray nozzle *E* in form of a veil of spray of conical form, or rather of umbrella shape. A syphon connected with the interior of the tower conducts any excess of liquid directly into the tower.

The filling device *A* and the spray nozzle *E* are so designed in relation to each other that when the spraying begins the circle which is being sprayed with liquid (or the solid angle of the spray cone) becomes gradually wider while the level of the liquid in the vessel *D* rises. When the liquid in the vessel *D* has reached its highest level and begins to fall again the solid angle of the spray becomes smaller again.

The apparatus is adjustable in such a way that between each period of action there is a shorter or longer interval, or it may be so adjusted that in the moment when the vessel *D* is completely emptied the vessel *A* gives off to *D* the liquid which had accumulated in it in the meantime, so that the spraying process begins anew. In this way the liquid spray changes its conical form continually as though an umbrella was opened and closed successively at regular intervals.

The liquid passes from a reservoir to the filling device *A* through the pipe *B* in a continuous stream. After a certain quantity of liquid has accumulated it is suddenly given off through *C* to the vessel *D*, which then begins to spray through *E* as described before.

This apparatus can be designed for any hourly quantity of liquid and for any tower cross-section, but for the present 5 cu. m. of liquid per hour and a tower diameter of 5 m are the maximum values for which it is built.

Mr. J. W. Sittig, Temple Court Building, New York City, is the sole American agent of this as well as the many other products of the Friedrichsfeld chemical stoneware factory.

Carborundum Refractories.

"Carborundum firesand," one of the by-products of the manufacture of carborundum in the electric furnace, has found very extended use in the last years as a refractory. From a bulletin, entitled "Carborundum Refractories" and issued by the Carborundum Company, of Niagara Falls, N. Y., the following information is extracted.

Chemically carborundum firesand consists of several silico-carbides of compositions ranging from $\text{Si}_2\text{C}_2\text{O}$ to $\text{Si}_3\text{C}_4\text{O}$, representing the partial reduction of silica by carbon at the intense heat of the electric furnace. This is furnished in the form of finely ground sand and is mixed before using with a binding material such as silicate of soda or fire clay to give it proper cohesion when brought up to heat. A silicate of soda of 52 deg. Baumé is best adapted for this purpose; it must be diluted with a certain quantity of water before being added to the firesand. When clays are used, a crude Jersey fire clay or one of equal purity should be taken. Kaolin may also be used with good results. Prepared fire clay, such as is furnished for laying up fire brick, is generally mixed with ground calcined clay or powdered fire brick, and is not suitable for the purpose.

In estimating the amount of material required for a lining of given dimensions, the weight of the carborundum firesand mixture may be computed as 85 lb. per cubic foot.

Lining of Crucible Brass Furnaces.—For linings of crucible brass furnaces a mixture is used consisting of 70 per cent carborundum firesand, 15 ground fire clay, 8 silicate of soda of 52 deg. Baumé and 7 water.

The silicate of soda is first added to the water and thoroughly dissolved. This solution is then mixed with the clay and firesand until the material acquires the consistency of molding sand. The constituents when thoroughly incorporated form a plastic mixture which is easily molded and has sufficient cohesion when tamped into place to retain its shape until fired. A sheet-iron form is provided of the same diameter as the interior of the furnace, consisting of a sheet of iron bent in cylindrical form, so that after use it can be sprung inward and be easily withdrawn. The center may also be constructed of wood and in such case should be well greased so that it can be drawn without the lining adhering to it.

The mixture is rammed up solidly between the center and the outside shell of the furnace, after which the center may be at once removed and the lining dried out with a slow wood fire. It can then be immediately put in service. In cold weather it is well to warm all the materials before mixing.

It is the practice in some foundries to provide channels or vents to assist in drying out the lining. This is done by placing four or five wooden strips about 2 in. by 1 in., spaced equally around the furnace inside and against the shell. These are withdrawn after tapping in lining.

This same mixture is used for lining ladles for brass, copper or iron.

Linings of Tilting and Rotary Oil-Burning Furnaces.—For tilting and rotary brass furnaces, in which the linings are subjected to the severest possible conditions, a mixture of 86 per cent carborundum firesand, 7 silicate of soda at 52 deg.

Baumé and 7 water is recommended. In many cases it is found advantageous to replace about 15 per cent of the firesand with an equivalent amount of kaolin.

This mixture is thoroughly incorporated and tamped between the shell of the furnace and a strong center provided for the purpose. When completed the center is carefully removed and the lining dried out at once with a slow wood fire. It is then gradually brought up to heat by means of the oil flame and can at once be put in service.

Many users of this style of furnace have found it practicable to throw in a small quantity of broken glass when first heating up the lining. This glass, when melted down, can be evenly distributed over the surface of the lining by tilting the furnace, and the interior thus acquires a smoother glazed surface which adds to the durability of the lining.

Patches can easily be made in places where the lining becomes broken or worn. It is best to cut out the broken parts entirely through to the shell, undercutting the old lining, in order that the new portion may be keyed into the old lining. For the purpose of repairs the same mixture is used as when putting in a new lining.

Repair of Firebrick Linings.—A mixture consisting of equal parts of carborundum firesand and fireclay or kaolin is extensively used for patching holes or broken parts of fire brick linings and for filling open joints in furnace walls. It is used to good advantage in ordinary crucible furnaces for brass and steel, also in copper reverberatory furnaces for keeping the side walls in repair. By its careful use it is possible to double the life of any firebrick lining. The mixture is largely used in repairing the Schwartz down-draft furnace, and also as a wash applied daily after the melting. Another efficient mixture for this purpose is the one recommended for crucible brass furnace linings given above.

A mixture of equal parts of firesand and fireclay is used successfully in repairing the brick linings of Tropenas converters, which are subjected to extreme corrosion by the flame.

Carborundum Furnace Paint.—Carborundum firesand may be made up in form of a paint or furnace wash by mixing 65 per cent carborundum firesand with 20 ground fine clay and 15 silicate of soda of 52 deg. Baumé and adding sufficient water to bring it to the consistency of a thick paint, which is applied to the surface of the lining with a broom or stiff brush. Two or three applications gave a proper coating. This makes an excellent protective coating for furnaces subjected to intense flame action and high temperature. It is preferably applied to new brick work, as it does not adhere so efficiently after the linings have become glazed over or covered with slag.

Special applications of carborundum paint are in arch and bridge walls of boiler furnaces which are subjected to severe flame action, in potters' kilns, welding furnaces and malleable iron furnaces. It is of great value in oil furnaces in protecting the brick from the well-known pitting and eating action of the high-pressure oil flame. Carborundum paint is used with good results as a cupola wash in protecting the melting zone of foundry cupolas.

Carborundum mortar is made from a mixture of equal parts of carborundum firesand and fireclay. It is used in laying up firebrick, in patching open joints and holes in brickwork and in general repairs. The life of a firebrick structure is generally limited by the life of the joints. A mortar of carborundum firesand is found to effectively stop the slagging and corroding at the joints.

Carborundum firebrick are furnished in the regular size, 9 in. x 4½ in. x 2½ in., and also in many of the special arch and key shapes. Many special shapes and forms are also made according to specifications and drawings. Carborundum brick have shown great economy in oil furnace work of all descriptions, in connection with the melting of crucible steel in forging and welding furnaces, in electric furnaces and in brass furnaces. They are, moreover, the only brick which will stand the intense heat generated in furnaces using powdered coal fuel.

A New Electric Recording Instrument.

A new type of electric recording instrument, particularly adaptable for use as a recording electric pyrometer, recording voltmeter or ammeter has been designed by the Brown Instrument Company, of Philadelphia.

Recognizing the necessity of a recording instrument which

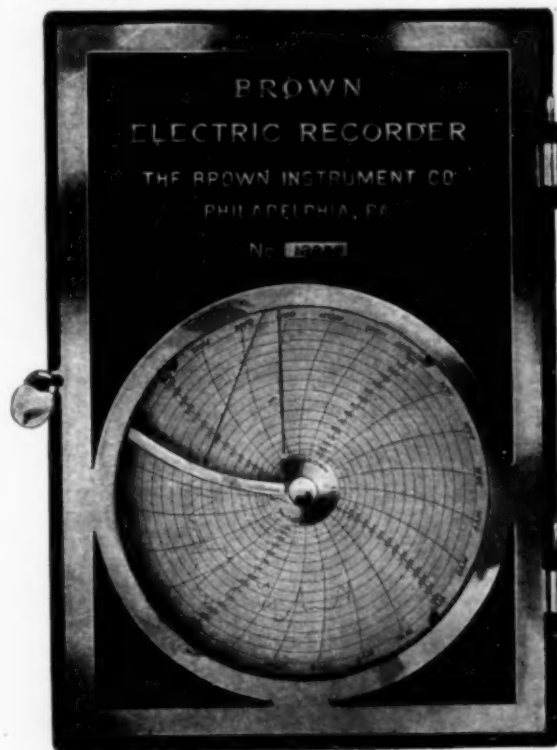


FIG. 1.—VIEW OF ELECTRIC RECORDING PYROMETER.

is both accurate in operation and simple in construction the Brown Instrument Company has spent much time in designing a recording instrument which can be mounted on the wall or switchboard and used by the most inexperienced workman.

In the common form of recording instruments no means are provided for shifting the recording pen arm out of the way when the chart which receives the record is changed, conse-

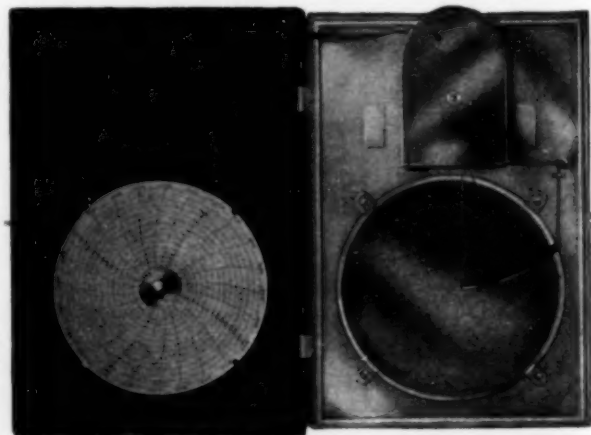


FIG. 2.—RECORDER OPEN.

quently there is serious danger of bending the pen in removing the chart. In this respect the new Brown recording instrument is most novel in construction, in that the essential parts of the instruments are mounted on the door instead of in the case. The clock mechanism and chart alone remain in the case when the door is opened. Consequently there is no possibility of

bending the pen because there is no occasion to handle it for any reason whatever. As soon as the door is thrown open the entire voltmeter system and the inking device is swung aside automatically, permitting the old chart to be easily removed and a new one substituted.

Fig. 1 (page 614) illustrates this recorder in operation with the door closed, while Fig. 2 shows the instrument with the door thrown open ready to have a new record chart installed.

The millivoltmeter system which is used in this new recording instrument is a simplified form of the D'Arsonval system which is the only system used in electrical instruments of precision. Also the recording pen comes in contact with the paper momentarily only for the purpose of making a dot, thereby eliminating all friction between the pen and the chart.

An inking pad is placed immediately beside the pen carrying sufficient ink for a week's supply, and this inking pad touches the pen point frequently, keeping the point damp. When the door is closed the arm seen at the left of Fig. 2 and operated by the clock mechanism, comes in contact automatically with the inking device, and every half minute, or quarter minute if preferred, it pushes the inking pad away from the pen, per-

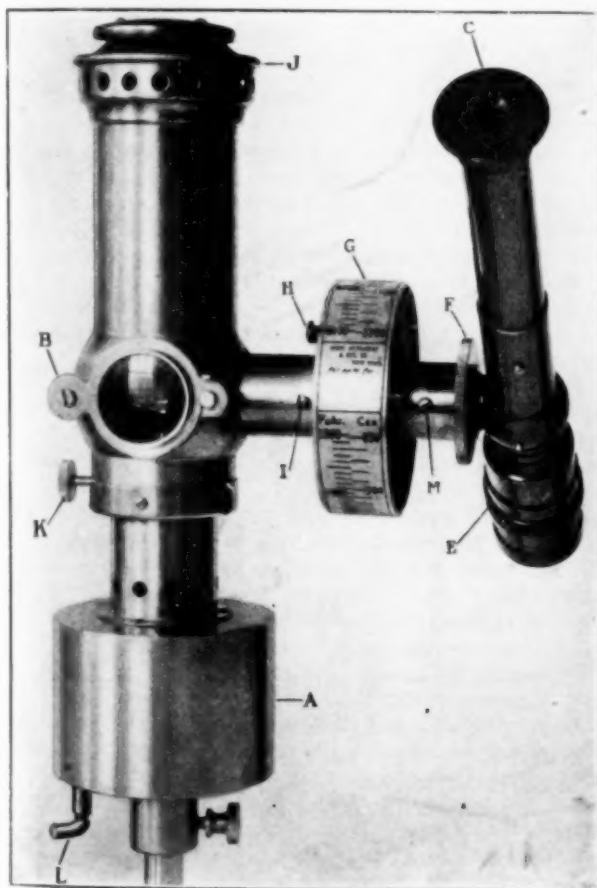


FIG. 1.—PYROSCOPE.

mitting it to swing freely. It then falls, allowing the inking pad by its own weight to press the pen against the paper.

This extraordinary simplicity of construction, the freedom from unnecessary parts, as well as the frictionless operation on the paper, will strongly appeal to those who understand the value or rather necessity of simple design for recording instruments.

Filter Papers.—We have received price list and samples of filter papers in sheets, rolls, circles, and folded-filters, made by Messrs. Carl Schlaicher & Schüll, filter-paper manufacturers of Düren, Rhenish Prussia. Messrs. Eimer & Amend of New York City are the sole agents for the United States.

A New Optical Pyrometer.

In many fields of metallurgy it is now clearly recognized that exact heat treatment is all-important and that the skilled eye of the most experienced attendant is not able to judge temperatures always correctly on account of the considerable variations of the illumination of the room with the weather, etc. This explains the rapid development of the pyrometer industry and the ready introduction of the pyrometer into com-



FIG. 2.—MEASUREMENT OF THE TEMPERATURE OF A TOOL BY PYROSCOPE.

mercial practice in recent years. Naturally the tendency in the design of various pyrometer types has been to develop robust fool-proof shop instruments from what were originally laboratory instruments requiring careful handling.

This same tendency is manifest in the latest design of an optical pyrometer, called the pyroscope and built by the Shore Instrument & Manufacturing Company of 555 West Twenty-second St., New York City. There is no necessity to deal here again with the underlying principle of all optical pyrometers. The chief points of the Shore pyroscope are its simple design, strong and rugged construction and comparative inexpensiveness. It is a little less accurate than a first-class and perfectly calibrated pyrometer, but is stated to give correct readings within 10 deg. Fahr.

The standard lamp used for comparison is a miniature kerosene flame. Ordinary kerosene is filled in the tank *A*, Fig. 1, the wick is lighted as in any ordinary lamp and the flame is adjusted to normal height (about $\frac{3}{4}$ in.) by the knob *K*. By now looking through eye-piece *C* at the hot tool, etc., which undergoes heat treatment, one gets a clear view as through any telescope. After focusing by turning the knurled ring *E*, the knob *F* is turned, which works the colored diaphragm and

controls the light intensity of the comparison reflector *R* shown in the special diagram in the upper left-hand corner of Fig. 2, which shows the image seen through the pyroscope. Knob *F* is turned until practically no difference can be seen between the redness of the reflector *R* and the tool under treatment. This can best be tested by going a shade higher or lower until balanced. The indicator *H* on the graduated drum *G* then gives directly the temperature reading.

By means of locking by the set screw *M* it is possible to set the instrument at a given temperature so that the work can all be heated up to a certain degree. The telescope can be used in any desired position on its pivot.

Recording Pressure Gauges.

The use of recorders has rapidly increased in recent years in connection with all kinds of measuring instruments, since a continuous record, furnished by a recording instrument, represents the best and most reliable control of the operation of a plant and the work of the attendants.

The Columbia pressure recording gauge, made by the Schaeffer & Budenberg Manufacturing Company, of Brooklyn, N. Y., and shown in the illustration, is noteworthy especially for



RECORDING PRESSURE GAUGE.

some special features of the record chart, namely uniform graduations, large effective record space and the division of the border into a black and a white section, denoting the day and night record respectively.

Columbia recording gauges are made in two sizes, suitable for 8-in. and 12-in. charts respectively and for pressures ranging from 4 in. water pressure up to the highest working pressure met with in practice.

Some of the applications of these gauges are the recording of steam, air, gas, water and ammonia pressures. These gauges may be had in dust and air-tight or moisture-proof case, and for temporary installations or field service portable gauges are provided.

The recording chart shows maximum and minimum pressure carried, the rate of rise or fall in pressure, in brief it is a complete log of the operation as indicated by the pressure at any time. A file of charts in the office of the superintendent enables him to compare the work done in the plant hour by

hour and day by day and to study in detail how the plant economy is affected by the pressure.

On the other hand the workman likes the recording gauge as it relieves him from the time-consuming and inconvenient writing down of the pressure gauge readings on the report sheet.

On the whole, the installation of pressure recording gauges leads to more conscientious operation and should prove a great incentive to improvements in all chemical works where gases or liquids are carried under pressure.

Titanium Steel.

The American Iron and Steel Association published recently the following statistics of the tonnage of various alloy steels during 1910.

Titanium steel	326,316	gross tons
Nickel steel	106,707	" "
Nickel-chrome steel	52,021	" "
Chrome steel	23,550	" "
Manganese steel	19,360	" "
Vanadium steel	9,049	" "
Other Alloys steel	30,816	" "

Total 567,819

As is pointed out in a recent circular of the Titanium Alloy Manufacturing Company of Pittsburgh, Pa., these statistics show that the output of titanium steel for 1910 was not only 300 per cent greater than the next largest nickel, but it was greater by 84,813 gross tons than the output of all the other alloy steels combined.

It is, of course, a question whether titanium steel can be considered as an alloy steel in the same sense as, for instance, nickel steel or chrome steel. In the case of the latter two, the nickel and the chromium are alloyed in certain proportions with the steel and give it certain valuable properties.

On the other hand, titanium was originally introduced in the steel industry as a cleaner. Titanium is an excellent deoxidizer and has also a very high affinity for nitrogen. It acts as a purifier of the steel, removing the gases from it, and improves greatly the structure of the steel. When acting in this way the titanium passes into the slag and if no excess of titanium is used, no titanium is contained later in the steel. Such titanium-treated steel is, therefore, not an alloy steel like nickel steel or chrome steel, but the results obtained by the purifying titanium treatment are so remarkable that they alone would explain the enormous increase of the use of ferrotitanium in the steel industry in the last few years.

However, more recently several investigators have found that the beneficial effects of titanium are not restricted to its action as a purifier but that titanium may give valuable properties to steel when the titanium is used in excess over that needed for purification alone. Facts illustrating this action have been cited to us from different investigators working quite independently of each other. It is to be hoped that exact information concerning this most interesting subject will soon become available.

The Non-Ferrous Metal Market.

Prices have fluctuated somewhat since our last report owing to conditions on the Stock Exchange and in foreign markets. The report of the Copper Producers' Association depressed the domestic copper market, as it seemed to indicate a contraction of business abroad. Tin fluctuated considerably as a result of both foreign and domestic conditions. Lead has eased off somewhat, while spelter has advanced.

Copper.—The latest quotations on this metal show a tendency toward lower prices, and there has been a large of-

fering on the part of Lake producers who wish to dispose of stocks before navigation closes. The quotation for Lake copper is $12\frac{3}{8}$ to $12\frac{3}{4}$ cents, with electrolytic at 12.05 to 12.10 cents.

Tin.—At the close of September tin could be bought at about 35 $\frac{5}{8}$ cents in New York, but as a result of the activity of the Syndicate abroad prices rose, and at present October tin is quoted at 41 cents.

Lead.—As a result of two successive reductions by the principal seller, this metal has fallen off about $\frac{1}{4}$ cent per pound, and there is little demand in view of the unsettled conditions. New York lead is quoted at 4.20 to 4.25 cents; St. Louis at 4.05 to 4.10 cents.

Spelter.—Prices have steadily advanced in view of large exportations and shortage in the domestic supply. Business has been on a heavier scale and consumers are buying in such quantities as can be supplied. Latest quotations were firm at 6 to 6.15 cents, New York, and 5.85 to 6 cents, St. Louis.

Other Metals.—Business in aluminium is somewhat improved, but prices are no higher. The metal is quoted at 18.5 to 19 cents, New York. Antimony is weaker and prices are reduced to 7 to 8.25 cents for various grades. Prices have fallen slightly in quicksilver; New York prices being \$45 per flask of 75 lb.; San Francisco, \$44.50.

Notes.

The **Kestner Evaporator Company**, which for a number of years has been located at 333 Walnut Street, Philadelphia, has recently removed its business to the Commonwealth Trust Building, Twelfth and Walnut Streets, in the same city.

Monel Metal Casting.—An interesting feat in the casting of Monel metal was recently accomplished in the foundry of the Bayonne Casting Company, Bayonne, N. J., where a four-blade propeller wheel, 16 feet in diameter, was cast in somewhat less than an hour. Fully 18,000 pounds of the metal were required to allow for gates and finish, the melting temperature being about 2500 Fahr. and the finished weight about 14,000 pounds. The wheel, because of its greater strength and freedom from corrosion, is to take the place of a steel wheel on the S. S. Madison, of the Old Dominion Line.

Mr. **H. M. Hirschberg**, 30 East Twentieth Street, has recently been appointed American representative of the Planawerke, Ratibor, Silesia, Germany, manufacturers of the well-known carbon electrodes bearing their name. There is such a big demand for products of this factory, that contracts are now being made for the latter part of 1912 and 1913 deliveries, Mr. Hirschberg already having closed for almost the entire output of the factory available for this market. These electrodes are being made in sizes up to 600 millimeters (2 feet) square cross section for rectangular electrodes or 600 millimeters (2 feet) diameter for round electrodes and in lengths up to 2000 millimeters (6 feet 7 inches). It is especially noteworthy that these electrodes are provided with a screw end so that new length can be added when required, thus doing away with all waste of carbon ends.

A New Departure in Handling Gritty Water.—For many years the pulsometer has found favor for dredging work, coal mining, sand suction, etc., because it has no rubbing surfaces so that no difficulty is experienced in handling gritty water. To overcome the terrific cutting action of high velocity water containing very hard flinty matter, the Pulsometer Steam Pump Company, of 17 Battery Place, New York City, announces that pulsometers are now being cast by a special patented process wherein the shell, while of the usual cast iron construction to give ample rigidity, is completely lined on the inside with a surface of carborundum. Sand blast and other abrasive tests would seem to indicate that this lining will stand almost indefinitely the wear of water moving at high velocity and containing anywhere from 40 to 60 per cent of hardest material.

The **International Institute of Technography**, an association of patent experts and technical bibliographers, has been organized to make comprehensive prior art examinations of patents and literature by a carefully organized system. A careful compilation of any single prior art from all the records, by an individual, requires weeks or months. Such laborious and expensive compilations have already been made, however, in comparatively every art, covering each to at least a comparatively recent period. The association has made it its business to know of, to compile, and to classify all such compilations, to ascertain where they are to be found, and to have a qualified expert at hand to examine them. The offices of the Institute are at 50 Church street, New York City.

The **Aero Pulverizer Company** announce that they have made an arrangement with the Cockburn Company, with works at Jersey City and offices at 126 Liberty Street, New York, to take over the manufacture and sale of the Aero pulverizer. The Cockburn Company has always manufactured these machines for the Aero Pulverizer Company, so that there will be no change in the product or workmanship. Mr. Wm. A. Evans will have supervision of manufacture and sales.

The **Copper Extraction Company**, Anaconda, Mont., has placed with the Ruggles-Coles Engineering Company, of New York, orders for two class "A 2," one class "B 10" and one class "F 8" dryers, for drying copper tailings.

Thermometers and Other Instruments for Breweries.—That chemical engineering plays an important part in brewing was well illustrated at the American Exposition of Brewing Machinery, Materials and Products, which will be held at the Chicago Coliseum, Oct. 12 to 22. Prominent among the exhibitors was the Schaeffer & Budenberg Manufacturing Company, of Brooklyn, N. Y., showing a complete line of its high-grade engineering instruments, among them its new recording thermometers, its special features being the mercury-filled steel capillary tube and the night and day chart, both new departures in thermometer construction. A full line of its brewers' thermometers, together with steam and ammonia gages for indicating and recording purposes, was shown, also its new steam engine and ammonia compressor indicators. Messrs. W. H. Shenton and T. J. Mentin officiated at their booth.

Two varieties of natural graphite are generally recognized in the trade—crystalline, or that possessing a lamellar, flaky or fibrous structure, and amorphous, including all other forms. Domestic flake graphite is replacing to some extent the Ceylon graphite used in the manufacture of crucibles for the metal industries, and the increased demand is being met by increased domestic production rather than by increased importation.

Obituary.

Horace H. Emrich, manager of the copper refinery of the Kyshtim Mining Works Company, of Kyshtim, Russia, was killed, according to newspaper cable dispatches, on October 17th, while standing at the window of his residence, by a gun shot from the outside. The Russian government has ordered an investigation. Mr. Emrich is a well-known member of the American Electrochemical Society and has here many friends who will be greatly shocked by the news of his sudden death. He was formerly superintendent of the copper refinery of the American Smelting and Refining Company at Maurer, N. J. In a letter which we received from him this summer, he wrote that he had just finished the design of and was erecting a modern steam generating plant for his refinery. The concluding words of his letter were: "Please give my regards to my friends and tell them that I do not find Russia such a fierce place to live in as we Americans are led to believe. In fact my wife and I find Kyshtim living, people, and climate all that could be desired, and we have spent a very pleasant year here."

Personal

Mr. Philip Argall, manager of Stratton's Independence, Ltd., sailed from New York on Oct. 18 to attend the annual meeting of his company. Mr. Argall will spend some time traveling abroad before his return to this country.

Mr. C. A. Dart has resigned his position as manager of the Rambler Copper & Platinum Company, Holmes, Wyo., and is now in Denver.

Mr. R. R. Foster, late superintendent of the metallurgical plants of the La Tula Mining Company, at La Luz, Guanajuato, Mexico, resigned his position to enter consulting practice. He will have headquarters at Torreon, Coahuila, Mexico.

Dr. Charles H. Fulton is now in charge of the mining and metallurgical department of Case School of Applied Science at Cleveland, Ohio. Dr. Fulton recently resigned his position as president of the South Dakota School of Mines, at Rapid City, in order to relieve himself of executive duties and devote more time to professional work.

Mr. Charles W. Goodale, of Butte, Mont., attended the meeting of the American Institute of Mining Engineers at San Francisco last month.

Mr. Franklin Guiterman, Colorado manager for the American Smelting & Refining Company, returned to Denver Oct. 1, after spending the summer in Europe studying the possibilities of electric zinc smelting.

Mr. Victor G. Hills, of Denver, Colo., who has been in Nova Scotia for some months developing a scheelite property, returned to Colorado for a short time last month. He was called as an expert in a mining suit at Telluride.

Mr. E. M. Johnson is now superintendent of the Altoona Zinc Works, at Altoona, Kan., having lately resigned his position at Depue, Ill.

Mr. Henry P. Lowe, managing director of the Frontenac Mines, Ltd., Central City district, Colo., has gone to England to attend the annual meeting of his company.

Mr. W. J. Quigley, manager of the Chihuahua Mining Company, Chihuahua, Mexico, spent last month in Denver.

Mr. P. E. Van Saun, of the Mill & Smelter Engineering Company, New York, is in New Mexico on professional business for his company.

Mr. Charles T. Van Winkle has resigned his position as superintendent of the Magna plant of the Utah Copper Company, Garfield, Utah, and has opened an office as consulting engineer in Salt Lake City.

Mr. A. W. Warwick, who has been in New York making tests on the Clancy cyanide process, has returned to Colorado, and is at Victor superintending the opening of the Colburn mill in which the Clancy process is to be used.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

473,186, April 19, 1892, Parker C. Choate, Brooklyn, N. Y. Zinc. Treats complex ores containing zinc, iron, sulfur, lead, gold, silver, antimony, arsenic, selenium, cadmium, bismuth, etc. The ore is crushed and, if it contains an excess of sulfur, roasted at a moderate temperature. It is then mixed with coal-screenings, and roasted on a perforated grate beneath which air is admitted. The zinc and other bodies volatilized therewith, e.g., oxides of antimony, arsenic, selenium, cadmium and bismuth, are caught in bags. The condensed fume is taken from the bags and roasted at a temperature of from 300 to 800° F. in a muffle-furnace, to drive off the impurities more volatile than lead or zinc. The zinc oxide is dissolved in sulfuric acid, the

lead sulfate settling out. The zinc sulfate solution is electrolyzed between anodes of lead or carbon and cathodes of zinc or copper. The freed acid rises to the surface of the cell and is drawn off through overflow pipes. As it contains considerable zinc, a portion is returned to the lixiviating tank and regenerated by the addition of fresh zinc oxide. Another portion is returned to the cell. Water is added from time to time to replace the loss. The deposited zinc is stripped off the cathodes and melted, or is melted therewith when zinc cathodes are used. The residual sulfates from the bottom of the lixiviating tank are smelted. The tailings from the roaster are also smelted to recover the iron, gold and silver.

484,869, Oct. 25, 1892, George J. Atkins, of London, England.

Gold, silver, etc. The apparatus illustrated is substantially the same as that of the patentee's previous patent 473,105. The amalgamating apparatus is connected to the cathode of the electrolytic apparatus or to the positive pole of a dynamo, and a supplemental anode is placed in the exit mouth of the amalgamating apparatus to cause direct deposition of any gold and silver in the solution. The process is claimed as a continuous treatment consisting in passing the ore through an anode compartment while in contact with the anode and then amalgamating it; or, after the anode treatment, passing it slowly through the cathode compartment and thence to the amalgamator.

495,212, Apr. 11, 1893, John Franklin Wiswell, of West Medford, Mass., Assignor to the Electric Gold and Silver Chlorination Company, of Kittery, Maine, and Boston, Mass.

Amalgamates precious metals by adding to the ore in a crusher a solution consisting of mercuric chloride and sodium hypochlorite. The mercuric chloride is prepared by passing a current through cups containing mercury immersed in common salt solution, carbon cathodes being used. The overflow from this tank is electrolyzed, carbon electrodes being used, to form sodium hypochlorite. The calomel formed in the cups is heated in a separate receptacle with aqua regia to form soluble mercuric chloride. Iron must be present during amalgamation, and if not present in the ore may be liberated from the iron crusher by the sodium hypochlorite, or may be introduced as iron filings.

BOOK REVIEWS.

Metallurgy of Tin. By HENRY LOUIS, D. Sc. 8vo: 138 pages; 41 illustrations. Price, \$2.00. New York: McGraw-Hill Book Company.

A reprint of a monograph by the author in "Mineral Industry," 1896, with such more recent information as was available. The author deplores the non-progressive character of the industry, also the profoundly secretive attitude of tin smelters, and pungently remarks: "To what extent these two facts, the slow advance of the industry and the secretive habits of those engaged in it, are related as effect and cause, may be left to the judgment of the reader."

Hendricks' Commercial Register of the United States for Buyers and Sellers. Twentieth annual edition. 1420 pages. Price, \$10. New York: Samuel E. Hendricks Company.

This well-known commercial register for buyers and sellers, which covers all branches of architectural, engineering, electrical, mechanical, and kindred trades and professions, appeared first in 1891. The twentieth edition, just issued, is more complete than any of the former.

The total number of classifications in the volume is now over 45,000, each representing the manufacturers or dealers of some machine, tool, specialty or material required in the above named industries. The whole represents upward of 350,000 names and addresses.

The new edition will, undoubtedly, prove as useful and reliable as the former ones.